REQ. NO. 3-9005

Energy, Mines and Resources Canada
Énergie, Mines et Ressources Canada

Research and Technology
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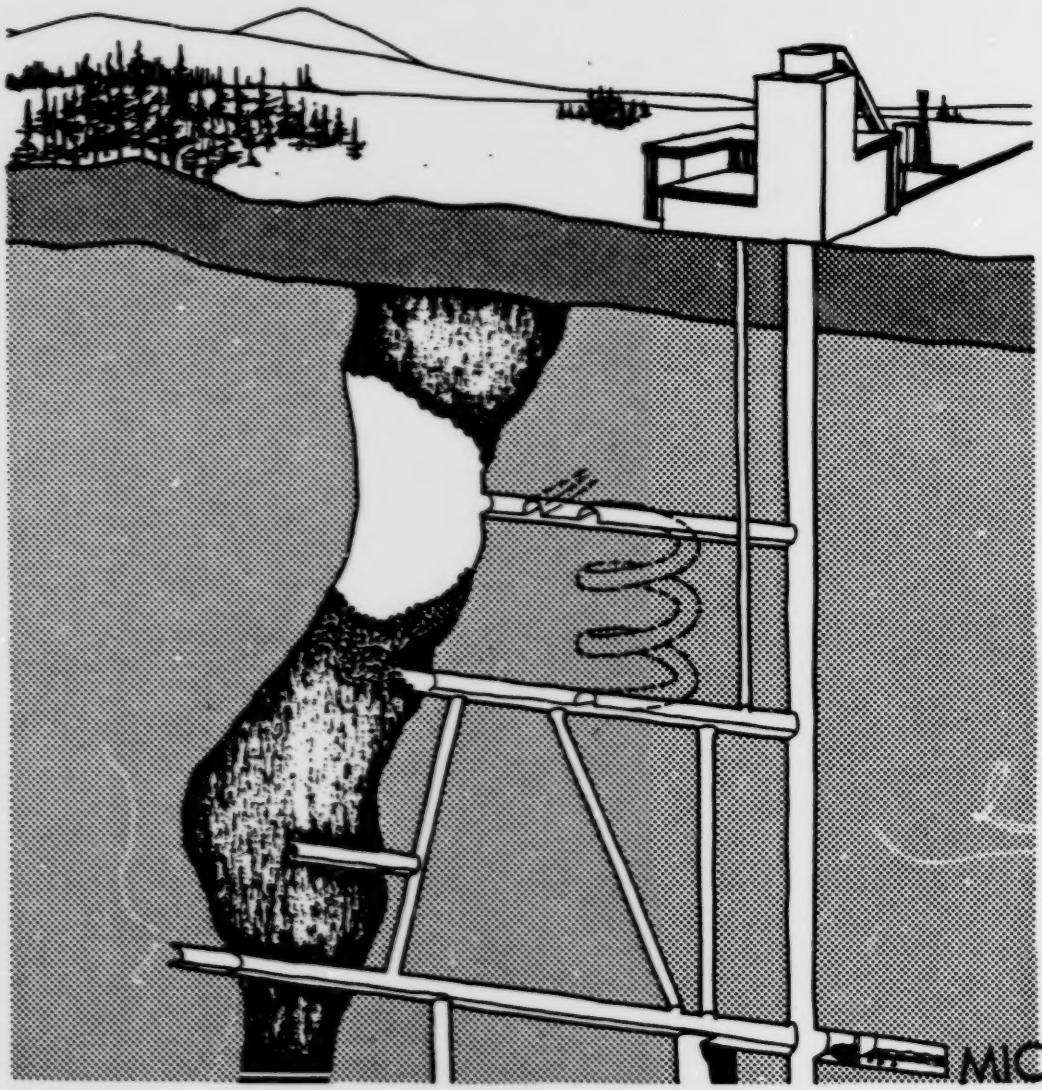
CONTRACT REPORT NO. 15Q83-00013

SURFACE PILLARS



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275 pp

FINAL REPORT

Energy, Mines and Resources Canada
Research and Technology

Énergie, Mines et Ressources Canada
Recherche et Technologie

FINAL REPORT

STATE-OF-THE-ART DESIGN AND STABILITY OF SURFACE PILLARS

(Ce rapport est disponible en français)

FEBRUARY 1984	YOUR FILE : # 26SQ.23440-3-9005 OUR FILE : # 3176-0000

27579

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I- INTRODUCTION

There exists a large number of underground mines in the province of Quebec and across Canada in which the mineralization occurs near the earth's surface or beneath a relatively thick layer of marshland, body of water or permanent surface structure. In such cases, a sufficiently strong section of rock must be left to protect the underground stopes from surface material incapable of supporting itself.

The recent disaster at the Belmoral mine in northwest Quebec reminds us of the importance of this concept. Those engaged in the mining industry have expressed a willingness to evaluate the technology developed to date related to the design of these sections of ore, or surface pillars, and to the evaluation and monitoring of their stability.

Our preliminary analysis revealed that surprisingly few studies have been conducted specifically on this type of pillar in Quebec, Canada or elsewhere.

1.1 MANDATE AND OBJECTIVES OF THE STUDY

The consulting firm Roche Ltd. has been retained by the Canada Centre for Mineral and Energy Technology (CANMET) to study the present situation regarding the design and stability of surface pillars in Canadian underground mines. This project was realized in collaboration with the Association des Mines de Métaux du Québec (AMMQ), the Centre de Recherches Minérales du Québec (C.R.M.) and representatives of Quebec mining companies.

The objectives of the study consist of the following:

- to evaluate, by means of an exhaustive review of existing literature, what has been realized in this field, emphasizing aspects related to surface pillar stability, design, regulations and technology.

- to prepare a series of case histories based on visits to six representative mines.

- to evaluate the possibility of developing a classification system for surface pillars and to determine general design criteria based on data obtained during the course of visits.

- to define the research requirements aimed at the development of more efficient pillar design, support and control techniques, emphasizing recent developments in applied rock mechanics and other relevant technological fields.

In order to fulfil these objectives, we have based our approach on three principal sources of information. The first and certainly most complex source was scientific literature on this subject. This bibliographic research constituted a background to which the knowledge of the operators was integrated. This data, which was obtained during the course of mine visits, constitutes the second source of information. Third, we relied to a great extent on the experience and competence of each member of the study team.

1.2 STUDY TEAM

In order to conduct this study, we considered it essential to work in collaboration with representatives of the industry and mining associations. Consequently, an advisory committee was formed to present ideas on the subject and to direct the study toward the most relevant areas. The sectors represented by the members of this committee are illustrated in figure 1.1.

The two consultants who were members of the study team were responsible for overseeing the general progress of the project, participating in the development of the study program and supervising the preparation of reports. Each consultant placed special emphasis on his particular field of specialization.

Several members of the personnel of Roche Ltd. were involved in various phases of the project.

1.3 FINAL REPORT

The final report on surface pillar design and stability follows a series of progress reports submitted upon completion of each of the three principal phases of the study which consisted of a preliminary analysis, bibliographic research and mines visits.

The final report is presented as follows:

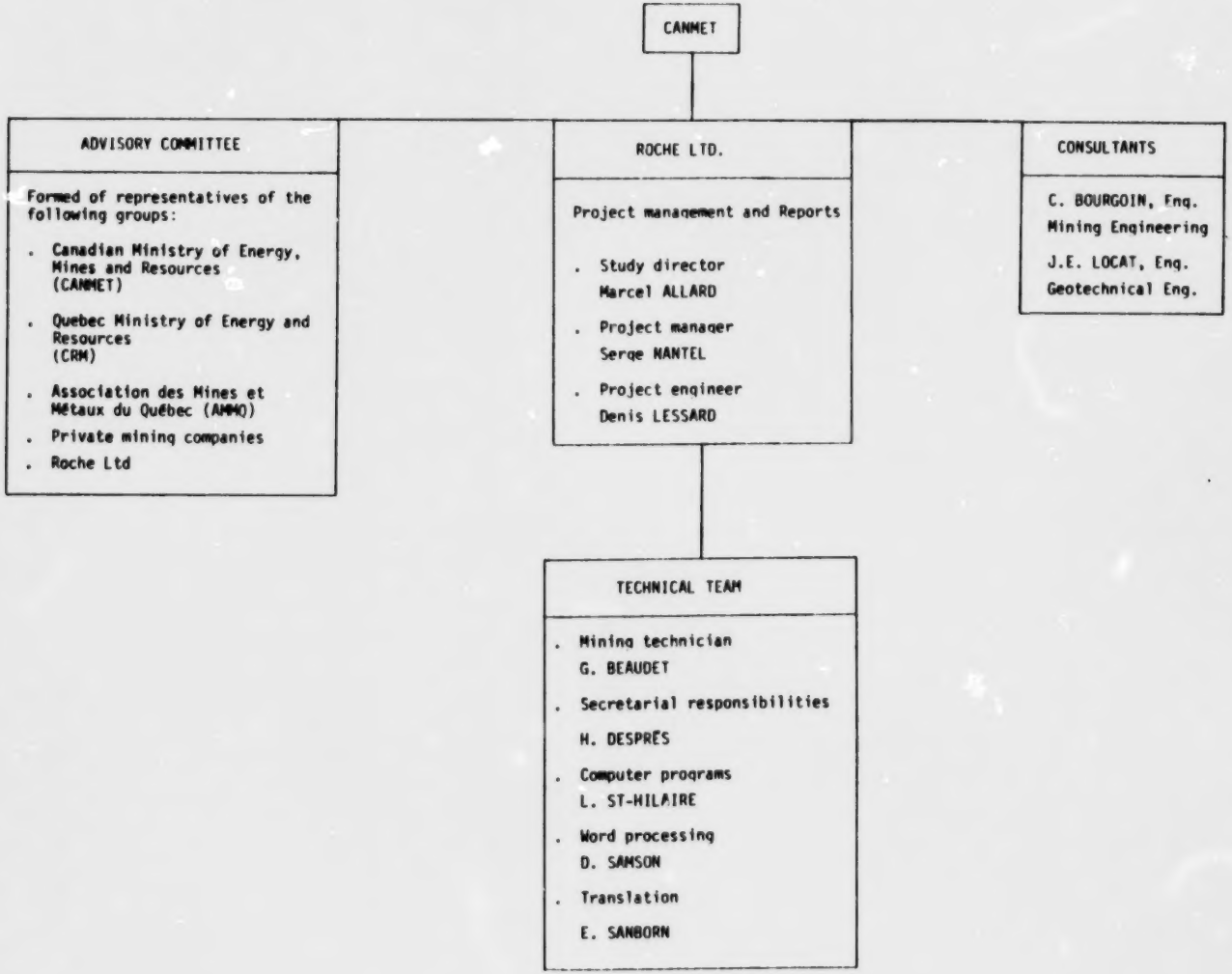

FIG. 1.1 ORGANIZATION CHART OF THE SURFACE PILLAR PROJECT

I- INTRODUCTION

In this section, the problem is examined, the mandate defined and the study team presented.

II- TERMINOLOGY

The principal terms used in literature and in the mining industry to designate the various types of mine pillars, including surface pillars, are defined in this section. If no definition was found for a certain term, we proposed a definition which appeared to be satisfactory to the principal participants in the study.

III- BIBLIOGRAPHIC RESEARCH

The most relevant data obtained from existing literature are discussed in this section. The product of the research is a bibliographic list of the works consulted, which are assigned a code indicating the contents of each reference.

IV- MINE VISITS

This section presents a table illustrating the types of activities regarding surface pillars conducted at each of the six mines visited. We have attempted to underline the unique aspects of each mine.

V- DISCUSSIONS

Following the first four sections, which are descriptive rather than analytical, this section constitutes a synthesis of the activities being carried out at the mines and the methods and techniques discussed in the literature. The mines visited are examined in a more general

context and the advantages and disadvantages of the techniques employed at these mines to design and control the stability of surface pillars are underlined.

VI- RECOMMENDATIONS

In this section, we present a list recommendations regarding measures which should be taken in order to increase the efficiency of surface pillar design. These recommendations are based on the most useful elements encountered in both the literature reviewed and at the mines visited.

VII- APPLICATION OF THE RECOMMENDATIONS

We have proposed an order of importance for the application of the study's recommendations.

ANNEX 1

Annex 1 constitutes a synthesis of the terminology used in the literature consulted and in the mines visited.

ANNEX 2

Annex 2 contains details regarding the bibliographic research method used, and inventory of the articles and a coded bibliography.

ANNEX 3

Six case histories are presented in Annex 3, each of which follows a common format.

II TERMINOLOGY

Before proceeding any further, we feel we should immediately define the term " surface pillar" . During the course of our literature review, we observed a virtual absence of terms used to designate surface pillars. In fact, the large majority of works consulted made no reference to this type of pillar, describing only those which support the floor and roof of underground stopes. In English literature, frequent reference is made to " crown pillars" , the horizontal pillars left in the upper part of stopes. By extension, it is the first horizontal pillar at the uppermost level of the mine which presently interests us.

European (French) literature is more specific thas English literature for it refers to " stot" , a zone of rock, mineralized or not, left around an opening in order to ensure stability. However, no direct reference is made to the term surface pillar.

As in the case of the literature consulted, the personnel of the mines visited does not use uniform terminology to refer to surface pillars. However, the terms " surface pillar" and " pilier de surface" were used at four of the six mines visited, as shown in table 1.1 presented in annex 1. Of the many terms used to refer to pillars, the following expressions are the most common:

pilier de surface - surface pillar
pilier de couronne - crown pillar
stot de protection

The mine personnel often uses these terms interchangeably, in both French and English. Since there is no standardization among the mining companies we have proposed the following term and definition:

surface pillar: a rock mass of variable geometry, mineralized or not, situated above the upper stope(s) of the mine, which serves to permanently or temporarily ensure the stability of surface elements.

A simplified surface pillar is illustrated in figure 2.1. This definition was submitted to and discussed with the advisory committee who agreed to the necessity of clearly defining the terms. We proposed the use of the term " surface pillar" and our definition to the personnel of the mines visited. The definition appeared to conform to the reality that they intuitively defined as a surface pillar. The notion of stability and safety implied by this term was unanimously accepted.

For information purposes, we have defined and illustrated the other types of mine pillars encountered during the course of this study in Annex 1.

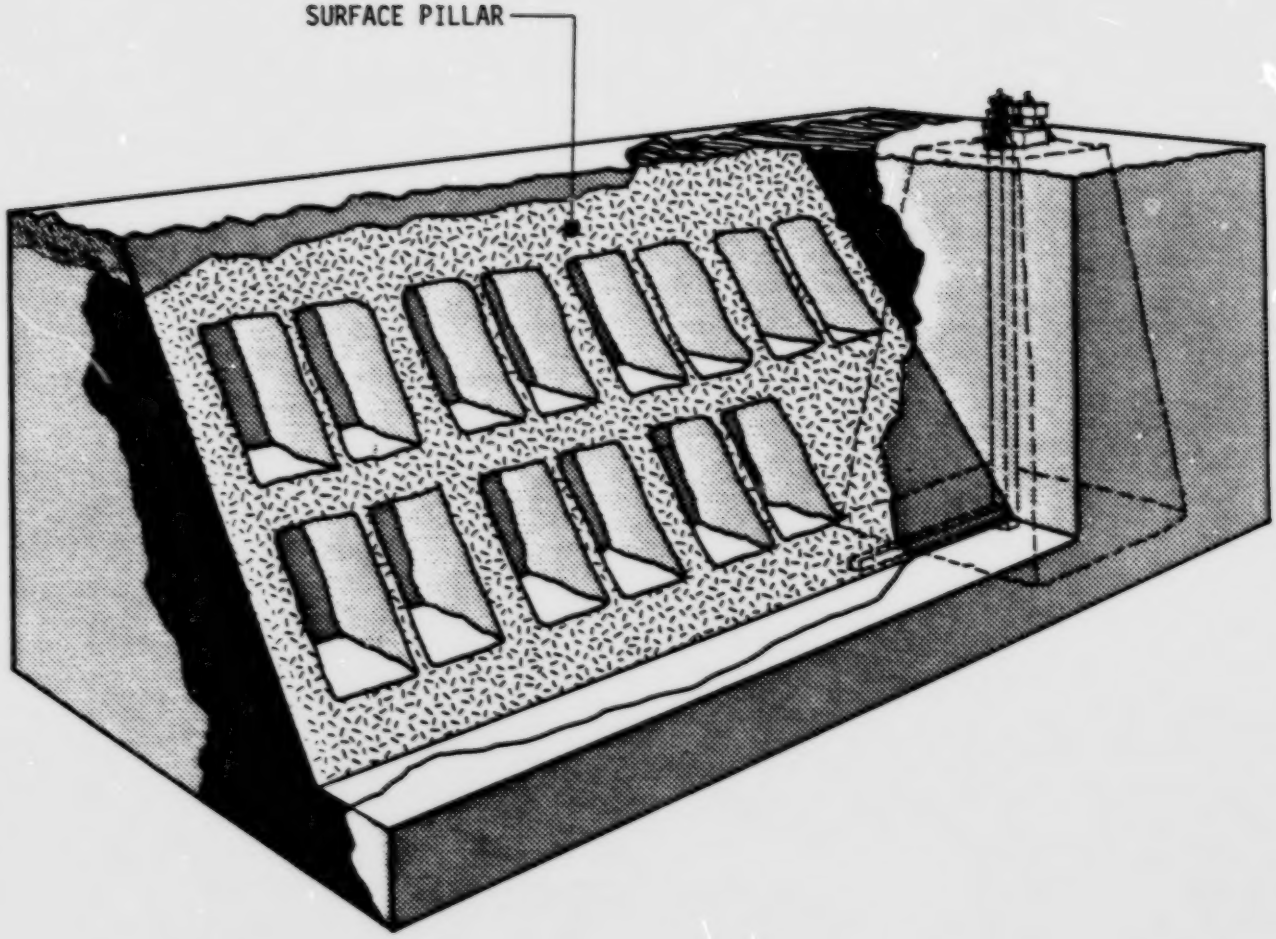

FIG. 2.1 SURFACE PILLAR

III- BIBLIOGRAPHIC RESEARCH

As previously mentioned, the objective of conducting bibliographic research was to compile a data bank regarding the design and control of surface pillars.

A conventional research approach was adopted based on resources from local scientific libraries and information from North American computer data bank. The use of data bank proved to be particularly useful for they offer fast and efficient access to large amounts of information which would otherwise have been impossible.

Our bibliographic research consisted of the following steps:

- familiarization with the terms used
- selection of key words
- research of relevant works
- obtaining documents
- classification of articles
- preparation of a bibliography
- coding of documents

These steps are described in further detail in Annex 2. We would like simply to point out that the product of our research is a coded bibliography of documents. The code enables the content and quality of each document retained to be quickly determined. The reference listed below illustrates the advantage of using such a coding scheme.

REFERENCE:

3- ALLCOTT, G.A., ARCHIBALD, D.E.

Description of Pillar Behaviour at Heath Steele Mines, CIM Bulletin, Vol. 74, no. 834, PP. 80 - 87.
1A, 2B, 3D, 4B, ---, 6A, 7B, ---, 9C, 10C, ---, ---, ---, ---, ---, ---.

By referring to table 2.3 in Annex 2, we can see that:

1A - This article discusses a Canadian topic

2B - This article is a case history

3D - There is nothing overlying this mine

4B - The failure criteria used are applicable to pillar in general

6A - The measurement of constraints and deformations of the pillars is discussed

7B - The evaluation of stability is based on an analytical method

9C - The deposit is flat and inclined at more than 45° with the horizontal

10C - Heath Steele is a metal mine

3.1 CLASSIFICATION OF PILLARS

Although no surface pillar classification exists in the literature studied, two classification techniques for vertical pillars which are commonly used in coal and metal mines were retained.

The first classification method is discussed by several authors including Hedley et al. (1979), Coates (1981) and Tincelin (1982). With this method, pillar strength is linked to factors such as the ratio of the smallest width to height, the volume of the pillar, the angle of major discontinuities to the axis of the pillar and the shape of the pillar (round, rectangular, etc.). This geometric approach is extensively used in the design of this type of pillar.

The second technique is based essentially on the optimization of knowledge regarding properties of the rock mass in which pillars will be formed. This classification is described by Hoek and Brown (1980), Bieniawski (1983, 1979, 1974) and Laubscher (1977). These authors provide tables or abacus which enable the classification of pillars as a function of the quality of the rock mass composing them. Due to technological advances, these classifications have become more and more highly sophisticated and can be used to solve practical problems. They also provide information regarding pillar stability, the type and amount of support required in particular cases, etc.

Such indications of rock mass behaviour are based on extensive information gathered in different types of mines or tunnels and on the practical experience of the researchers.

The principal geomechanics classifications are summarized in table 3.1. The authors are presented in an approximate chronological order.

This latter approach is interesting in that it can be applied repetitively, thus eliminating the subjectivity of evaluations to the greatest possible extent.

TABLE 3.1: GEOMECHANIC CLASSIFICATIONS (LEGAST, 1981: DISSERTATION ON BLOCK CAVING, McGILL UNIVERTITY, MONTREAL)

		1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	
(FROM CHRONOLOGICAL ORDER)		TERZAGHI •	LAUFFER •	ONODERA •	COATES & PARSONS	BURTON	DEERE et al	DEERE & MILLER •	HANDIN •	STAPLEDON •	BRIOLI •	GOODMAN	McMAHON et al	MÜELLER	WICKHAM et al	BIENIAWSKI (CSIR)	LAUBSCHER	BARTON et al (NGI)	TAYLOR	
DEFORMATION OF CHARACTERISTICS					X															
STRENGTH OF ROCK MATERIAL	UNIAXIAL COMPRESSIVE				X	X		X		X				X		X	X	STRESS INTER. FACTOR	X	
	DEGREE OF LOOSENING										X									
	MOD. OF ELASTICITY			X				X												
WEATHERING				X												X†	†		†	
DISCONTINUITIES	R.Q.D.		VAGUELY DEFINED				X						X			X	X	X	X	
	NO OF SETS					X									X		X	X	X	
	SPACING				X						X			X	X	X	X		X	
	CONTINUITY														X	X	•		•	
	CONDITION											X			X	X (1)	X	X	X	
	STRIKE / DIP ORIENTATION														X	X	†		†	
WATER INFLOW															X	X	X	X	X (2)	
STAND UP TIME			X																	
GROSS HOMOGENEITY					X															
LITHOLOGY					X				X						X					
ROCK LOAD		X															†	X	†	

SUMMARY OF PARAMETERS USED IN ROCK MASS CLASSIFICATIONS

- • Quoted by John
- † Adjustment
- (1) Joint separation, originally separate parameter
- (2) Included with joint condition
- • Implied

3.2 DESIGN METHODS

Approximately two-thirds of the articles retained discuss methods of evaluating overall pillar stability. Pillar dimensions are obtained by conducting detailed analyses based on various parameters. Stability represents the first condition which must be met in pillar design. Certain authors present design guides in which the various stages of pillar design are outlined. Duvall (1977) lists nine important principles; Gill (1981) presents an interactive approach (figure 3.1); Parker (1983) divides pillar design into four major phases; Pariseau (1975) introduces the notion of study costs into his discussion; and Brissenden (1980) examines pillar design in the broad sense of the term.

In all of these articles, there is one common notion that should be retained. The design methods proposed can be divided into the following three phases:

- Evaluation of the capacity and strength of the pillar
- Determination of the stress fields in the rock mass of the pillar rock mass of the pillar in question.
- Comparison of the two sets of values by means of equations.

Most of the authors discuss the traditional problem of rock mechanics which consists of transferring laboratory results obtained from small samples to actual rock mass properties. For example, Duvall (1977) examines shape effect and Tincelin and Riveline (1965) discuss the variation of test results as a function of the size of the sample and

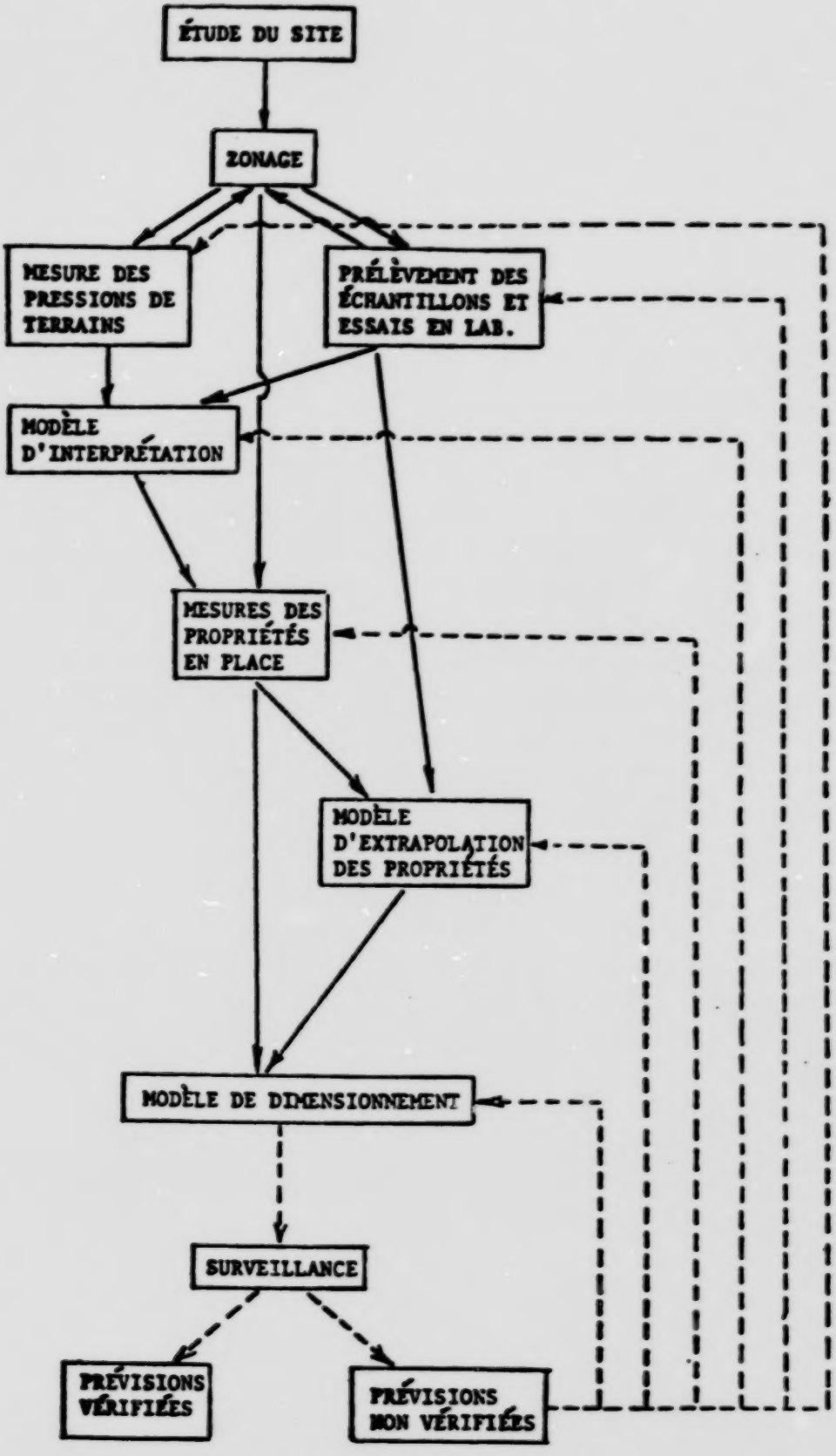

FIG. 3.1 DESIGN METHOD IN ROCK MECHANICS (Gill, 1981)

the negative variation of rock strength with time. This is particularly true in cases of non-elastic, non-homogeneous rock masses in which structure plays a dominant role in its stability.

The fact that perfectly homogeneous rock masses, which are easily quantifiable by means of a series of equations, are rarely encountered has given rise to a school of thought in which pillar evaluation is based on a set of properties or parameters which contribute unequally to the determination of its carrying capacity.

Before we go on, we should mention that in mines in which deposits of coal and oil shale occur, the more or less plastic nature of these materials appears to enable the use of simple relationships between laboratory results and the actual behaviour of the rock mass. This is not the case in the majority of metal mines and more particularly in deposits in which the mineralization occurs near the surface where rock masses are usually very fractured.

Page and Brennen (1983), Bieniawski (1983), Barton et al. (1974) Hoek and Brown (1980) and Duncan and Devane (1983) direct their approach towards an analysis of variables likely to contribute to defining the strength of these jointed rock masses as accurately as possible. These techniques, which are often based on civil engineering works, can be divided into two groups. The first group consists of geomechanics classifications based on the in situ analysis of the behaviour of a number of previously classified rock masses. Based on these analyses, the authors draw a set of conclusions regarding the stability of the rock masses.

The classification criteria are applied objectively and systematically in order to obtain a rock mass rating. This rating corresponds to a class characterized by specific properties and behaviour.

The second approach, that of Duncan and Devane (1983), Page and Brennen (1983) and Hoek and Brown (1980) among others, is based on the hypothesis that it is possible to adapt values obtained in a laboratory to the rock mass. Each of these authors propose several adjustment factors which will be discussed in greater detail in Annex 2.

The determination of stress fields in the pillars is often simplified to the greatest extent possible in analytical calculation methods. It is often a question of the tributary area theory according to which stresses are proportionately distributed on the pillars. Until recently, this technique was the most widespread. Today, however, there is a clear trend towards the use of numerical analysis techniques based on the finite-element method. Mathews et al. (1983) use this technique to obtain the distribution and size of stresses in rock masses as a function of the geometry of the stopes, their respective location and of the presence of major faults which modify the uniform theoretical field of these stresses. Hedley et al. (1979) used a similar technique for evaluating the stability of the Kidd Creek mine and the surface pillar left between the bottom of the former open pit and the underground mine. Approximately 10 articles dealing with the application of numerical analysis techniques were retained.

The next step in the design process consists of comparing the two elements which the authors attempted to define: the strength of the rock mass and the forces acting on it. Duval (1977) discusses the use of the beam theory whereas Hoek and Brown (1980) limit themselves to determining whether the strength of the rock mass is greater than, less than, or equal to the stresses acting at various points of it. Finally, advocates of geomechanics classifications evaluate stability by associating the rock mass in question with the known behaviour of a rock mass having comparable properties.

Unfortunately, we found very few cases in which these design methods were applied to surface pillars in metal mines. Case studies in this field have been conducted by Page and Brennen (1983) and Hanninen et al. (1973), for a horizontal sedimentary deposit. Southey (1969) presents a case history of an submarine iron mine in which the design of the surface pillar was based on very subjective criteria. Tincelin (1982) adopts a theoretical approach which attempts to explain various situations encountered in metal mines produced by room-and-pillar mining. Touseull and Rich (1980) document and analyze a massive rock failure and Hedley et al. (1979) describe rock mass behaviour at the Kidd Creek mine.

3.3 SAFETY FACTORS

The notion of safety factors was discussed in the literature consulted. It is a common pratice in the field of civil engineering to design a structure 1.2 to 1.5 stronger than actually required. In the mining sector, this notion is used to compensate for the ignorance of real caracteristics of rock mass.

Faced with the difficulty of defining the strength of a given rock mass with precision, several authors use the laboratory-obtained strength value divided by an correction factor, which is chosen according to the conditions of ground. Duvall (1977), for example, used a factor of between 4 and 8 for tensile strength and between 2 and 4 for compressive strength. Goodman et al. (1980), even after having applied adjustment factors related to the pillar shape and size to the laboratory value, divide the value thus obtained by a factor " f" , which is variable with the conditions of ground.

In general, a security factor could be used as a compensation tool to fill the missing data from rock mass analysis. This security factor notion is given more in details in annex 2, and tends to show that the S.F. does not really represent the degree of confidence given to this kind of work.

3.4 DESIGN EQUATIONS

Extensive documentation exists regarding mine pillars, particularly those of coal mines, which link the strength of a pillar to its shape. These design equations, as we have chosen to call them are discussed in detail in Annex 2.

These equations are very practical and are generally presented as follows:

$$\text{strength of pillar} = \text{correlation factor} \times \frac{\text{width of pillar}}{\text{heigth of pillar}}$$

It is therefore relatively easy to determine the shape of pillars depending on the strength we would like them to have, as a function of the forces which apply at specific areas of the mine. Hedley et al. (1972) has proposed similar equations for pillars of metal mines.

However, no formula for determining the dimensions of surface pillars was provided in the works consulted. The equations discussed can only be used to evaluate the strength of a point in the rock mass as a function of the stresses acting at this point, as discussed in section 3.1. Consequently, a trial-and-error approach must be adopted to determine pillar dimensions or the beam or arch theory must be employed to define these parameters.

3.5 MONITORING PROGRAMS

Approximately 20% of the articles retained dealt specifically with pillar deformation and another 20% discussed deformations is general.

Within the framework of monitoring programs, extensometers or linear potentiometers are installed in order to detect wall, roof and floor displacement (Thian, 1983; Alexander et al., 1981; Barrett et al., 1979; Hedley and Wilson, 1975).

In addition to enabling the measurement of deformations and the control of stability, instruments can serve to verify theoretical data predicted prior to mine production.

The follow-up deformations in a given stope allows to observe the evolution of the local conditions and failures which may occur in the near future may be foresseen. Load cells can be installed in strategic locations in order to determine in situ stresses.

Barrientos and Parker (1974), who have analyzed data obtained from a dozen pillars equipped with instruments, state that stresses measured in partially robbed pillars are less than theoretically calculated values. They explain that this is due to the arch effect in which stresses are transferred to the abutments of the stopes.

Virtually the only instruments used with surface pillars are designed to measure surface subsidence. Tincelin (1982), Singh (1979), Dejean and Arcamone (1979) and Orchard (1969) have investigated or evaluated surface movement caused by underground mining operations. This interest in surface movement is a consequence of physical constraints such as surface structures, dams and bodies of water.

Another method of monitoring underground stopes is based on the use of geophysical instruments to detect microseisms or noises produced by rock mass failures (Leighton, 1953; Gibbons, 1980; Hedley et al., 1979). The recording of noises in the rock mass can assist in preventing mining accidents; the amplitude and frequency of microseisms are parameters which can be used to evaluate stope stability, to detect critical zones and to determine the necessary corrective measures which should be taken.

3.6 REGULATIONS

One of the objectives of our mandate was to review existing literature regarding legislation specifically applicable to mine pillars. During the course of our bibliographic research, we came across a document into the Belmoral mine tragedy and safety conditions in underground mines. In the third volume entitled " La sécurité dans les mines souterraines" , an entire chapter was devoted to this question.

The following two paragraphs are taken from Chapter 2 (page 51) of this document and deal with the introduction of regulations in effect in Canadian and American mines.

"In pursuing our objectives, the commission considered it essential to conduct a study of mining legislation and regulations in effect in Quebec, the rest of Canada and the United States. The study only examines regulations which are currently in effect applicable to underground metal and non-metal mines.

The major aspects analyzed are as follows: surface pillars, mining operations beneath aquiferous zones or bodies of water, working alone, underground lighting, communications with the surface, exploration of the surface, emergency exits, access to stopes, auto-evacuation, safety zones, ground control, training of miners, and haulage drifts. The form, content and coercive power of the regulations were of particular interest to us. This study shows that the many mining regulations applicable to underground mines have certain common characteristics."

One of the four characteristics common to all of the regulations should be mentioned:

"The third characteristic of mining regulations consists of the total absence of explicit standards regarding certain engineering-related aspects of mining activity such as surface pillar design in underground mines... (p. 51, Commission, volume 3, 1982)."

"The commission's study clearly shows that in the majority of regulations examined, special cases are taken into consideration and described as if it were impossible to establish standards applicable to the entire mining industry" (p. 51, Commission, volume 3, 1982).

Riddle (1980) examines the regulations of the Surface Mining Control and Reclamation Act of United States which deals with problems caused by subsidence in considerable detail. However, because his analysis only pertains to coal mines, it is of no use to us within the framework of this study.

Many other authors discuss mining guidelines (Orchard, 1969; Hedley et al., 1979; Bieniawski, 1983) and others summarize mining regulations (Skelly and Loy, 1976) but always in the field of coal mining.

It appears that no regulations currently exist which could serve as guidelines or provide information to operators of metal mines regarding surface pillar design.

IV- MINE VISITS

A subtantial amount of information was gathered regarding surface pillars during our mine visits. Although this information is not of equal interest from one mine to the next, it often deals with the same aspects. In order to summarize the practices carried out at each mine, we have prepared several tables. Not wishing to judge the relevance or degree of thorouhness of the studies conducted, we have weighted the works realized at each mine. This weighting system does not take the specific requirements of each mine into consideration. For instance, one mining company may require detailed information regarding the nature and behaviour of the overburden overlying the surface pillar whereas the priority of another company may be to acquire extensive knowledge of the hydrogeologic cycle.

The weighting technique should enable the reader to select the mine report which will provide the greatest amount of information regarding a specific aspect. The symbols used in the following table are as follows:

- ○ little detail
- ◑ moderately detailed studies
- ● very detailed studies
- ▬ no study realized / not applicable

Table 4.1 summarizes the various studies and analyses conducted regarding surface pillars in the six mines visited. It should be noted that only one of the mines visited is currently at the surface pillar design stage. The others terminated the production of the upper stopes long ago.

The substantial amount of data gathered during our mine visits is described in detail in Annex 3. This data can be classified as follows:

- thirty sets of geometric values related to the dimensions of existing pillars which seem to illustrate that a two-to-one ratio (minimum) exists between the thickness and width of stable pillars. The ratio of thickness to width of pillars which have collapsed (mine No. 5) is 1:1. These values are summarized in table 4.2.

- a general survey of mining methods previously used in the design of surface pillars and the evaluation of their stability. These methods generally involve the operators' experience acquired during the mining of zones which have properties similar to those of the surface pillars in question.

- a summary of the methods used to verify and ensure long term pillar stability. These include routine visual inspections, filling of the stopes, and the use of artificial support under the surface pillar. In the mines visited, few instruments have been installed.

- a description of the most frequently used methods for determining the properties of the material overlying the mineral deposit, the location of the contact of soil and rock and the groundwater level.

TABLE 4.1 SUMMARY OF STUDIES AND ANALYSES REGARDING SURFACE PILLARS

STUDY	MINE NUMBER 1	2	3	4	5	6
HYDROGEOLOGY						
. Visual evaluation	○	-	-	-	○	○
. Specific analysis	-	◑	◑	●	-	-
SURFACE CONDITION						
. Soil removal	-	-	◑	-	-	-
. Drainage	-	◑	◑	●	-	-
. Drill holes	-	◑	◑	●	-	○
. Laboratory analyses	◑	◑	○	●	-	-
. Drawings and plans	○	◑	◑	●	○	○
. Seismic surveys	-	-	-	○	-	◑
MODELLING, SIMULATION						
. Computer model	-	-	◑	◑	-	◑
. Simulation	-	-	-	-	-	-
ROCK MECHANICS						
. Internal program	◑	◑	◑	●	◑	◑
. Outside studies	-	○	◑	○	-	◑
. Laboratory testing	-	-	○	○	-	-
SURFACE PILLAR SUPPORT TECHNIQUES						
. Conventional bolting	-	◑	◑	-	◑	◑
. Grouted rebars	-	◑	●	-	-	-
. Resin-anchored rebars	-	-	-	-	-	●
. Loose fill	-	-	-	-	-	-
. Hydraulic fill	-	●	-	-	-	-
. Consolidated fill	-	-	◑	-	-	●
. Grouted cables	-	-	-	-	-	●
. Shotcrete	-	◑	-	-	-	-
INSTRUMENTATION, STABILITY CONTROL						
. Measurement of displacements	○	-	-	●	-	●
. In situ measurement of stress	-	-	-	-	-	○
. Visual inspection	○	◑	○	○	○	○
. Systematic monitoring	◑	●	●	-	-	-
ROCK MASS EVALUATION						
. Integrated classification	●	○	○	◑	-	○
. Type of rock	-	◑	-	-	-	-
. Structure	-	-	○	◑	◑	○
. R.Q.D.	-	-	○	-	◑	●

TABLE 4.2: GEOMETRY OF SURFACE PILLARS

SURFACE PILLARS	MINE NO. 2							MINE NO. 3									MINE NO. 4				MINE NO. 5						MINE NO. 6			
	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30
Length (m)	61	46	61	107	76	84	79	38	61	30	43	40	30	-	37	-	30	48	22	22	50	56	58	58	59	58	250	250	250	250
Width (m)	5	21	9	9	12	5	6	2	15	6	6	4	3	3	5	3	22	22	15	15	17	18	15	18	17	21	40	40	45	40
Thickness (m)	59	49	49	49	58	17	56	8	12	9	14	6	8	9	21	23	35	35	35	35	14	14	14	18	38	25	179	179	179	179
Height (underlying stope) (m)	46	46	46	43	40	55	50	46	91	107	-	-	-	-	-	-	20	20	20	20	37	37	37	37	37	37	24	24	24	24
Ratio: thickness/ width	12	2	5	5	5	3	9	4	0.8	2	2	2	3	3	4	8	2	2	2	2	0.8	0.8	0.9	1	2	1	4	4	4	4
COMMENTS:																														
Lake at surface (thickness) (m)	1	12	11	11	12	-	2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	13	13	13	13
Overburden (thickness) (m)	3	5	6	6	5	7	3	14	3	-	-	-	-	-	-	-	45	45	45	45	8	8	14	8	8	8	19	19	19	19
Mines residues (thickness) (m)	-	-	-	-	-	2	-	-	-	-	4	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Marshland + Overburden (thickness) (m)	-	-	-	-	-	-	-	-	-	4	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Collapsed pillar*	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	*	*	*	*	*	*	-	-	-	-
Overburden removed*	-	-	-	-	-	-	-	-	*	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-

- relevant information regarding the application of rock mass evaluation techniques. Mine no. 1 clearly illustrates the utility of conducting such an evaluation in order to gather data on the rock mass properties.

- a systematic approach to surface pillar design encompassing an analysis of the majority of relevant parameters: hydrogeology, support techniques, mining method, calculation of stability, model simulation, etc. This type of approach was adopted by mine no. 4.

The ideal design process for surface pillars or any other element related to mining operations can be divided into five major stages:

1. the observation stage during which basic data is gathered. The problem is defined in this phase in order to determine the number and nature of elements required to clearly define the subject under study;

2. the second phase encompasses all operations related to measurements which serve to quantify important parameters. During the course of this phase, laboratory and in situ tests are conducted in order to determine the mechanical properties of the rock mass and stresses acting in proximity to it; several elements of the initial observation are also specified in greater detail;

3. geometric data are then analyzed together with the measurements obtained in order to detect inaccurate or inconsistent points and to establish the relationship between these two types of data;

4. in the design stage, all data collected are integrated to form a uniform whole. The ultimate objectives of this phase are to determine how certain operations or excavation works can be carried out in light of the existing stresses, and how to act on the area and to understand the forces at work in order to be able to weight our findings;

5. the primary objective of the last phase consists of verifying whether the results of in situ and laboratory testing correspond to predicted results. During this monitoring phase, necessary corrective measures are taken.

As previously mentioned, not all of the mines visited followed this procedure relatively rigourously. One of the most obvious conclusions of this part of the study is that a much greater amount of data and number of measurements were available from the mine site than through mathematical calculations.

With respect to the application of this design process, the mines visited were not entirely successful. Although they gathered a considerable amount of field data, they were unable to integrate them in a coherent whole. With the exception of mine No. 4, none of the mines provides a specific procedure for surface pillar design.

We would have liked to have seen the mines keep specific files on surface pillars. Unfortunately, this was not undertaken at the mines visited. It appears to us that the transition from observation to scientific design of surface pillars was overlooked.

Although this omission does not often affect the stability of the surface pillar, it makes it impossible for the operators to determine the exact value of the safety factors or the variables which were taken into consideration in the decision to leave a pillar of a particular shape. We feel that the best results can be obtained by combining these two elements, that is, that the knowledge and experience acquired during the production of a mineral deposit should serve to orient stability studies of a more theoretical or mathematical nature.

Each of these five design stages are summarized in Annex 3. Special emphasis is given to the methods used, the techniques developed in particular situations and the steps followed to obtain the desired results.

V- DISCUSSIONS ON SURFACE PILLARS

According to our definition of surface pillar, all underground mines, with the exception of mines produced by means of block caving, have at least one surface pillar. Indeed, we know of no mine which operates without a roof. However, for a number of reasons, all mines do not have to be concerned with the design and optimization of their pillars. The fundamental concern is to be able to recognize the moment at which surface pillar design should be seriously considered.

One of the ways in which this can be achieved is to classify surface pillar so that mine operators can select a specific category which corresponds to his own pillar. Other methods consist of the realization of detailed studies of local conditions and the application of regulations which would force the mines to take action.

5.1 CLASSIFICATION

The objective of classifying surface pillars is to provide a simple but reliable method of evaluation which allows operators to assess their situation relative to other known cases. Unfortunately, although classifications have been developed for rock masses and pillars in general, there exists no such classification for surface pillars.

The mining industry does not have to develop classifications to operate safely; they simply ask themselves whether their surface pillar(s) are stable. This explains the reason for which terms used to describe pillars can be summarized as stable, partially stable or unstable pillars. The mining companies have not developed classification which would allow them to more accurately answer the question

of surface pillar stability and to more easily assess their situation.

Canadian mines are neither behind nor ahead of the underground metal mines of other countries.

The classification technique for pillars of vertical mines is generally based on two approaches:

. the determination of the shape and dimensions of pillars

. the evaluation of various properties of the rock mass by means of a systematic method which is called geomechanics classification.

In section III, we saw how vertical pillars may be classified by means of one of these approaches. In the case of surface pillars, we do not feel that we have enough examples to attempt to employ a geometric classification technique which could eventually be used in the design of these pillars; too many parameters exist which vary from one mine, and from one zone, to the next. Consequently, only very vague conclusions can be drawn from table 4.2 since there is no way of linking the properties of each rock mass. However, we have made the following observations:

. The ratio of thickness to width of the thirty pillars studied ranged from 0.8 to 12.

. The ratio of thickness to width of the six pillars which collapsed at mine no. 5 ranged from 0.8 to 2.0, with an average of approximately 1.

. 84% of the pillars studied and judged to be stable had ratios of 1 to 5.

This is all that is revealed by these geometric relationship as is the case for all approaches limited to obtaining pillar dimensions.

A more complete and scientific approach would be to develop a surface pillar classification based on the evaluation of a set of parameters typically related to these pillars and which directly affect their stability. For example, the rock mass could be evaluated by means of a geomechanics classification (the presence of surface water has negative effects on the final evaluation, the addition of artificial support systems has positive effects, etc.).

Such a classification is outlined in figure 5.1. Il should be noted that the weighting or percentage of points assigned to each parameter will be defined. This classification is not original in that it was developed based on the works of authors of previously mentioned geomechanics classifications.

Nevertheless, it is the first classification which attempts to objectively establish a link among surface pillars and which could eventually be used in their design.

FIGURE 5.1 SURFACE PILLAR EVALUATION

PARAMETERS EVALUATED	POINTS ASSIGNED
1) Rock mass quality (geomechanics classification)	---
2) Geometric relationships (thickness width, length)	---
3) Mining method and sequence (geometry of excavations)	---
4) Support method (filling, bolting)	---
5) Nature and thickness of soil overlying the deposit	---
6) Topography of bedrock	---
7) Presence of water (hydrogeological conditions)	---
8) Surface works	---
9) Neighbouring underground workings, in operation or abandoned	---
Rating example	1 000 (maximum)

The 1 000 points could, for example, be divided into 10 classes of 100 points; the higher the rating, the greater the stability of the surface pillar.

0--100--200-300-400-500--600-700-800-900-1 000 (points)

↑ Surface pillar on the verge of collapse

↑ Stable rock mass without openings (before mining)

5.2 DESIGN

The works reviewed offered a wide choice of design approaches which included the parameters to be analyzed or compared in order to obtain logical, comprehensive results. The authors provide equations, nomographs or behaviour models on which the design of stable stopes is based. However, these approaches must be adapted before they are applied to surface pillars since they were developed for the vertical pillars of coal and metal mines or for civil engineering works.

With the exception of mine no. 4, none of the mines visited provided an example of a precise, well-structured approach to surface pillar design. However, they did express a willingness to ensure underground stability and safety, and a desire to take economic factors into consideration in the design of surface pillars.

The approach taken by the mines in the past can be summarized as follows: a conservative pillar thickness was determined by means of a general evaluation of the nature of the rock constituting the surface pillar. This evaluation was normally based on the analysis of diamond drilled holes or the comparison of the conditions and properties of similar rock masses. Then, mining proceeded from the bottom of the deposit, advancing upwards. The behaviour of the roof was evaluated as the mining operations progressed and ground stability was monitored. The decision to terminate mining operations was based on the judgement of the mine personnel or on the detection of signs of instability.

Today, these same mines, particularly mines no. 2 and no. 3, monitor the stability of old pillars. Few mathematical techniques were used in the design of these pillars. However, the experience of the mine operators and the progressive method employed to mine the upper part of the mineral deposits are valuable factors which should not be overlooked. It could be very useful to the mining industry if this knowledge was discussed a various conference and meetings. The present study represent a first step in this way.

This approach is still used today, as in the case of mine no. 4, with the distinction that the existing conditions of the mine are evaluated in greater detail. A description of the methods used at mine no. 4 has been included in Annex 3. Essentially, the mine personnel has attempted to evaluate the situation by means of mathematical techniques, and has adopted the cut-and-fill stoping method in order to control ground stability. They have even simulated the collapse of the roof in order to evaluate the behaviour of the overburden should failure of the surface pillar occur.

We feel that this is a very safe approach to surface pillar design. It must be realized that the acquisition of detailed knowledge regarding the parameters or factors which may influence surface pillar design requires considerable time and money, without which it is futile to consider the optimization of surface pillar design.

We observed that there is a direct relationship between surface pillar design and the mining method used to form the pillar. Goodman (1980) illustrates this relationship by defining the effective area of a vertical pillar as the real area of the pillar less the surrounding fractured and altered zone, which is the result of both the mining method used and time.

Consequently, it is desirable that the design of surface pillars be realized by at least two specialists, a mining engineer and a geomechanics engineer.

5.3 INSTRUMENTATION, CONTROL, SUPPORT

The primary objectives of developing monitoring programs based on the use of equipment or on systematic inspections of the stopes are to detect any design deficiencies, to verify certain predicted behaviour, and to determine the effects of modifying the initial design. The wide range of measuring instruments available is described in a considerable number of works and their application is illustrated by practical examples. The instruments which can be used in the design of surface pillars can be divided into four categories:

- instruments designed to measure displacement, e.g. extensometers, optical readers ...
- instruments designed to measure stress, e.g. flat jacks, triaxial load cells ...
- detectors, seismic detectors and microseismic instruments
- instruments designed to measure water pressure, e.g. piezometers

Although the majority of these instruments have been used in specific instances at the mines visited, the operators depend to a much greater extent on systematic visual inspection of the stopes. We feel that this is an necessary process which would be pursued. However,

we also believe that it is essential to maintain at least one system of extensometers in operation after the data required for pillar design has been gathered. These instruments would enable the detection of the earliest signs the deformation which could lead to pillar failure. The location and position of these instruments must be carefully selected depending on local structural conditions.

With regard to surface pillar support systems, we observed that the mines visited are using grouted or resin-anchored steel rebars to a greater and greater extent. This technique conforms to the theory that the use of active support systems preserves the initial strength of the rock mass to the greatest degree possible. The rebars must, however, be installed as soon after the excavation of openings as possible, which is not always the case.

5.4 REGULATIONS

Since, the Belmoral mine tragedy, several organizations have expressed concerns regarding the importance of clarifying the question of surface pillar design in underground mines. The Mines Inspection Service of the provincial government, using the regulations regarding the safety and protection of workmen in mines and quarries established by order-in-council in virtue of the Quebec Mining Act has already taken action. The Service requires that mines touched by this problem conform to a series of measures outlined in a memo addressed to them. The memo also stated that a request for modifications would be submitted to the Occupational Health and Safety Commission (C.S.S.T.), which has had juridiction over mines since January 1981, in order to have these measures sanctioned.

The commission of inquiry investigating the Belmoral mine tragedy has studied the question and proposed three recommendations related to surface pillar stability. The first recommendation deals with the general stability of mine workings. The commission appears hesitant before the standardization and codification of design parameters (Commission, Volume 3, 1982) due to the extreme diversity of rock types and quality. In states that by imposing a method which includes all elements of a sound management and operation, mine safety will be ensured, mining companies will not be tied to the use static technology and the flexibility necessary to adapt to various conditions will be provided. Such a process would be much more effective than the stipulations of existing regulations.

The second recommendations deals with the exploration of the surface overlying future deposits. The commission identifies five types of overburden and proposes a minimum of exploration works to be conducted in order to determine the properties of the overburden as accurately as possible.

The third recommandation is more directly related to surface pillars and deals with short and long-term stability and pillar design.

A committee has recently been formed by the Occupational Health and Safety Commission to revise regulations regarding health and safety conditions in mines and quarries. Unfortunately it is too early to obtain any information regarding the nature of these revisions and to know whether or not the recommendations of the Mine Inspection Service and the commission of inquiry will be integrated into the new text.

The situation regarding regulations is therefore as stated is 1982 by the commission of inquiry (Volume 3, 1982). The inspectors of the Occupational Health and Safety Commission continue to use current regulations concerning mines as well as general powers granted to them by virtue of article 180 of the Occupational Health and Safety Act (LRQ, C.S.-2.1), paragraph 5 of article 180 as follows:

> "In order to ensure that a building, a structure or civil engineering works are stable, require the employer, principal contractor or owner to produce an attestation of solidity signed by an engineer or architect, or an attestation contemplated in section 54."

To our knowledge, no regulations related directly to the design and monitoring of surface pillars in metal mines exist. Furthermore, the term surface pillar does not appear in any existing legislation.

Having conducting a brief review of the available literative on this subject and observing the lack of regulations in this field, we strongly encourage the development of legislation directly applicable to surface pillars which would serve more is a set of guidelines than a set of restrictions.

We favour the approach proposed by the commission of inquiry and feel that the mining industry would be taking a considerable step forward by recognizing the paramount role played by mine operators in the safety of mines and stability of surface pillars.

5.5 CONCLUSION

It is extremely difficult to determine the consequences of developing openings in the non-homogeneous, non-isotropic and highly irregular environment which characterizes surface pillars. When the exact nature of a material is unknown, the most logical approach to adopt consists in using it with caution and in analyzing its behaviour under various conditions. This is what is done by the operators of the mines visited and what should be done by other operations.

We should qualify our statement by adding that with regard to the adoption of systematic design approaches such as those described in table 5.1, the mines visited have made some progress. However, as mentioned in section IV, the transition from the collection of data to scientific design is not made. The mines have become more interested in monitoring stability particularly by means of systematic visual inspection.

The actual condition at the mines reflects that encountered in our review of literature: no precise process has been developed for surface pillar design. Surface pillar design depends on the initiative of each company.

TABLE 5.1: DESIGN STAGES AND ELEMENTS

OBSERVATION

- Overburden
- Rock contact
- Relative position of openings (geometry)
- Borehole cores
- Water

MEASUREMENTS

- Rock mechanics properties
- Stresses
- Displacements
- Water flow

ANALYSIS

- Rock mass evaluation
- Weighting of in situ rock strength

DESIGN

- Experimental knowledge of the behaviour of similar rock masses
- Analytical models
- Simple theoretical calculations
- Drainage
- Support

MONITORING

- Instrumentation
- Inspection
- Simulation

VI- RECOMMENDATIONS

In light of the following factors, we have proposed a set of recommendations:

- very few case studies have been conducted or scientific texts written regarding surface pillar design;
- successefully employed rock mass evaluation techniques are available;
- it is possible to adopt mathematical techniques to the evaluation and design of pillars;
- it is essential to provide mining companies with an information handling system;
- design methods used by the mines are based on experience and a step-by-step method;
- three of the six mines visited no longer need to design surface pillars, as is the case for the majority of old mines, but will have to monitor the stability of existing pillars;
- there is a lack of uniformity in the design process;
- there are high costs associated with the realization of studies is this field.

The objective of these recommendations is to assist in the development of a methodology for the design of surface pillars. We believe that it is essential that a concensus be reached among the mines with regard to the design phases defined in section V so that a data bank can be developed to integrate information from each of the mines.

These recommendations favour the conducting of research to enable the development or optimization of certain elements within the five proposed design stapes.

6.1 OBSERVATION STAGE

The mines visited have the equipment available to obtain data regarding overburden, groundwater and surface water conditions, etc. Traditional drilling techniques, mapping, surveying and mine plans are used. Two areas of research appear to be important due to the costs involved.

For this reason, we propose the following:

. OPTIMIZE OVERBURDEN AND BEDROCK EVALUATION METHODS; determine the degree of precision required for the design of pillars and the best method to achieve this, evaluate costs.

. DEVELOP AND PROPOSE STANDARD PROCEDURES FOR THE EVALUATION OF BOREHOLE CORES. It is possible to obtain much more information from these cores than is currently being obtained.

6.2 MEASUREMENT STAGE

Technology is currently available for the determination of the mechanical properties of rock masses and natural stress fields. The only recommendation that we propose regarding this area is:

. DEVELOP A GEOMECHANICAL DATA BANK by an independent organization in which all values obtained from field and laboratory tests can be integrated together with all additional relevant data.

6.3 ANALYSIS

This is the most important and most subjective phase of the entire design process. It encompasses the question of how to group together all the data obtained in preceding steps into a logical whole. To grasp the relative significance of the various parameters, such as rock mass composition, structure, degree of fracture and alteration, is a veritable problem to which no easy solution exists. Some of the mine have decided to use relatively elaborate classification techniques (NGI, CSIR, R.Q.D.) whereas others seem to lack rigour in this area.

It is of paramount importance in this stage to be able to evaluate the carrying capacity of the rock mass. Ideally, all subjective evaluations should be eliminated. We propose the following:

. COMPARE AND EVALUATE THE MOST WIDELY KNOWN GEOMECHANICS CLASSIFICATION TECHNIQUE and determine their applicability to the conditions of the Canadian mining industry.

6.4 DESIGN STAGE

As previously mentioned, at the majority of mines visited, the design of openings and stopes is based on the personnel's knowledge of rock mass behavior. After the basic date is compiled, certain situations can be compared to similar conditions encountered elsewhere in the mine. This approach, based on experience, is certainly valid but does not allow an objective solution to questions regarding surface pillar stability.

The use of analytical models is slowly spreading in the mines. However, these models cannot provide sufficently satisfactory explorations. It is our opinion that several deficiencies must be corrected.

. DEVELOP BETTER ANALYTICAL MODELS. It would be useful to define the basic elements required to obtain valid results. A team of specialists in this field could evaluate existing models in the market and even develop a dynamic model which could be used by mine personnel.

. EXAMINE THE ENTIRE QUESTION OF DRAINAGE IN GREATER DETAIL. It could be useful to better understand the behavior of the water at the contact of soil and rock and in the rock mass in order to reduce its effects on overall stability.

. CONDUCT A GREATER AMOUNT OF RESEARCH IN THE FIELD OF GROUND SUPPORT. What type of support is best suited to the ground conditions? How and to what extent can the use of a certain support techniques increase surface pillar stability?

6.5 MONITORING

The monitoring of rock mass behaviour is considered essential by the mining companies visited and is used in order to detect all signs of instability. At the majority of mines, however, the monitoring process is neither highly developed nor systematic. The management of the mines largely depends on their employees to detect signs of instability and even this is done rather informally. Instruments are used to a small extent only. The monitoring of workings is perceived more as a preventive measure than a process of monitoring predicted behavior. Operators must be convinced of the utility of available instruments in particular situations.

We feel that the most valable recommendation with regard to the monitoring process is:

. THE MINE SHOULD BE USED AS A RESEARCH LABORATORY. Equip surface pillars with instruments and follow their progress over time. This particular mine site could serve as a model for other mines.

VII- APPLICATION OF RECOMMENDATIONS

Having outlined a set of recommendations, we propose that various organizations be entrusted with the responsibility of conducting the research in these fields or of contributing to the research activities.

In our opinion, the practical product of the research should be the publication of a DESIGN GUIDE explaining the stages essential to the rational and scientific design of surface pillars. At least one chapter of the guide should be devoted specifically to the monitoring of the stability of existing pillars and to the optimal mining method for extracting ore immediately beneath these pillars.

Such a guide would give priority to the Canadian mining industry, this guide could also form part of a more general guide related to the stability, in the broad sense of the term, of underground mines.

We would like to summarize our eight recommendations, this time in order of priority:

1. OPTIMIZE OVERBURDEN EVALUATION METHODS.

2. DEVELOP AND PROPOSE STANDARD PROCEDURES FOR THE EVALUATION OF BOREHOLE CORES (not the grade of ore but the rock properties).

3. COMPARE AND EVALUATE GEOMECHANICS CLASSIFICATION METHODS.

4. ESTABLISH A GEOMECHANICS DATA BANK for existing surface pillars.

5. DEFINE, DEVELOP AND ADAPT SUPPORT METHODS applicable to surface pillars.

6. CONSIDER THE DEVELOPMENT OF A DRAINAGE TECHNIQUE for the upper part of mineral deposits.

7. DEVELOP BETTER ANALYTICAL MODELS.

8. USE A MINE WHOSE SURFACE PILLAR IS REPRESENTATIVE AS A RESEARCH LABORATORY, where the design method would be verified.

The variables implied in these recommendations are presented in table 7.1.

In our opinion, the cost of research would be compensated by the advantages gained by the mining industry from such research. Furthermore, it would be worth examining the real cost in terms of human lives, capital and negative publicity the entire mining industry resulting from a major surface pillar collapse.

The proposed guide would be a working tool adopted to the needs of the mining industry. It would allow mine personnel to adopt an ideally optimized, efficient method which would provide reliable results at a minimum cost.

TABLE 7.1 DEVELOPMENT OF A SURFACE PILLAR DESIGN STRATEGY

STAGE	1 DISTRIBUTION OF REPORT ACCEPTABLE OF RECOMMENDATIONS	2 RESEARCH	3 GUIDE	4 VERIFICATION OF THE METHOD
MINING COMPANIES	. preparation of uniform files on surface pillars	. provide a site for an in situ laboratory . involvement in a specific field of research	. involvement in research depending on interests and possibilities	. in situ evaluation pillars by means of the guide
MINING ASSOCIATION	. reaching of a consensus on the five proposed design stages	----	----	. collection and evaluation of results obtained
PUBLIC ORGANIZATIONS	. financing to initiate the process	. initiation of research on topics discussed in recommendations 1, 2, 3, 4 and 5	. preparation of a surface pillar design guide . research on points discussed in recommendations 6 and 7	. testing of the design method and initiation of the applied research described in recommendation 8.
PRIVATE CONSULTING FIRMS	. promotion of project	. participation in research in various fields	. participation in the writing of the guide	. joint participation in the evaluation of surface pillars.

Moreover, the proposed program would be justified by the prevention of a single major collapse of a surface pillar. Finally, such an initiative would demonstrate that the mining industry wishes to continue developing production techniques and preventive measures to increase safety conditions in underground mines.

ANNEX 1

PILLAR TERMINOLOGY

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1.0 PILLAR TERMINOLOGY

During the course of our literature review, we came across various terms used to designate pillars which recurred in a number of articles. In order to clear up any confusion surrounding these terms, we have provided definitions and illustrations of them.

Three different groups of terms related to mine pillars emerged from our review. These groups are outlined in the following pages.

1.1 METAL MINE PILLARS

The first group consists of terms which are currently used in base metal mines. The form and location of the pillars listed below are illustrated in figure 1.1.

CROWN PILLAR	PILIER DE TOIT
ROOF PILLAR	PILIER DE COURONNE
LEVEL PILLAR	PILIER DE VOUTE
STRIKE PILLAR	STOT
HORIZONTAL PILLAR	

A pillar generally comprised of ore left temporarily in the upper part of the stopes to protect the stopes, drifts, extraction levels or access tunnels. It is generally removed once it has fulfilled its purpose. These terms contain the notion of stability. By extension, they are often used to define the protection pillar left on the surface of an orebody.

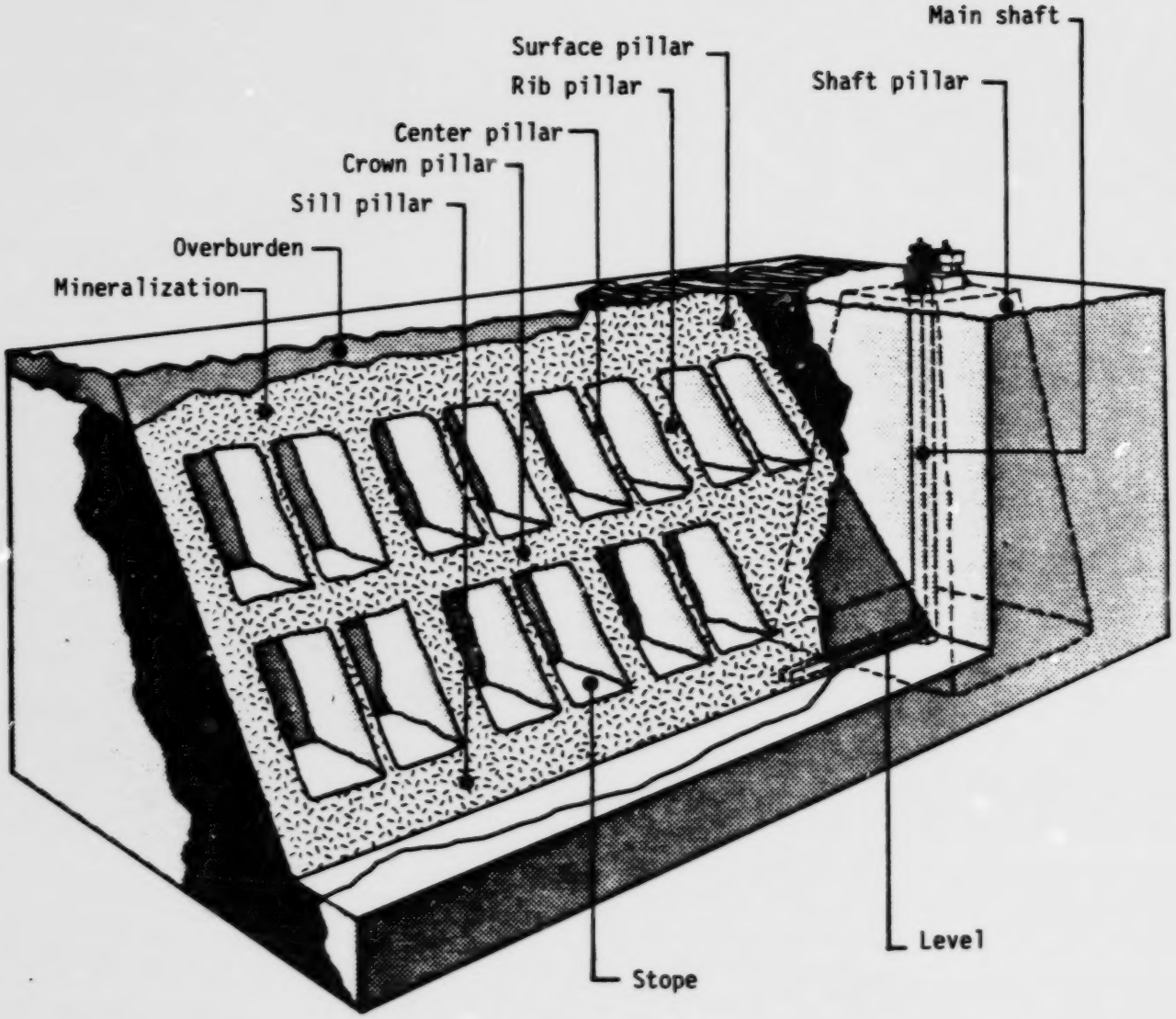

FIG. 1.1 METAL MINE PILLARS

RIB PILLAR — PILIER DE PAROI
DIP PILLAR — PILIER DE SÉPARATION
TRANSVERSE PILLAR

A pillar whose length is oriented in the direction of the dip of the orebody. This type of pillar is used in orebodies dipping at more than 25°. It joins the roof and floor of the mineralization and is generally monolithic, extending from one end of the mined-out zone to the other. Rib pillars separate the stopes from one another and can subsequently be worked.

SILL PILLAR — PILIER DE SOLE

Sill pillars are generally comprised of ore and contrary to crown pillars, are left standing temporarily between the sublevel and stope floor above to protect, drawpoints, access tunnels, etc. This type of pillar is horizontal.

CENTER PILLAR — PILIER CENTRAL

A pillar left in the center of a stope to reduce the span of the roof together with rib pillars. This type of pillar is often intersected by haulage drifts or access tunnels.

PILLAR — PILIER
STUB PILLAR — PILIER DE SOUCHE

A pillar cut in ore or waste rock in order to support the roof. These pillars vary in length, width, height and composition and can be either permanent or temporary. Used alone, the term pillar refers particularly to pillars formed by room-and-pillar mining in orebodies with horizontal or flat dip.

SHAFT PILLAR PILIER DE PUITS

A permanent pillar which is left to protect mine shafts during the entire life of the mine. The shaft and the pillar surrounding it may be either vertical or inclined. The shape of shaft pillar varies and can become wider at the base.

1.2 COAL MINE PILLARS

A second group of terms, illustrated in figure 1.2, appear to be used exclusively in the context of coal mines. Frequent reference is made to these terms in English literature. It should be noted that a number of reports discussing stability in coal mines have contributed to the development of metal mines, salt mines, etc. Coal mine operators have, in a certain sense, led the way for operators of other types of mines. The following terms are used to designate coal mine pillars:

BARRIER PILLAR PILIER DE SÉPARATION

Barrier pillars are left to separate the stopes or panels and to ensure roof stability. They also play an essential role in controlling underground ventilation and are generally permanent.

ENTRY PILLAR PILIER D'ENTRÉE

Located along stope and pillar development drifts (longwall mining) entry pillars are used for the production of coal veins. They also serve to protect access drifts to stopes, and are semi-permanent, in that they are used during the life of the sectors to which they give access.

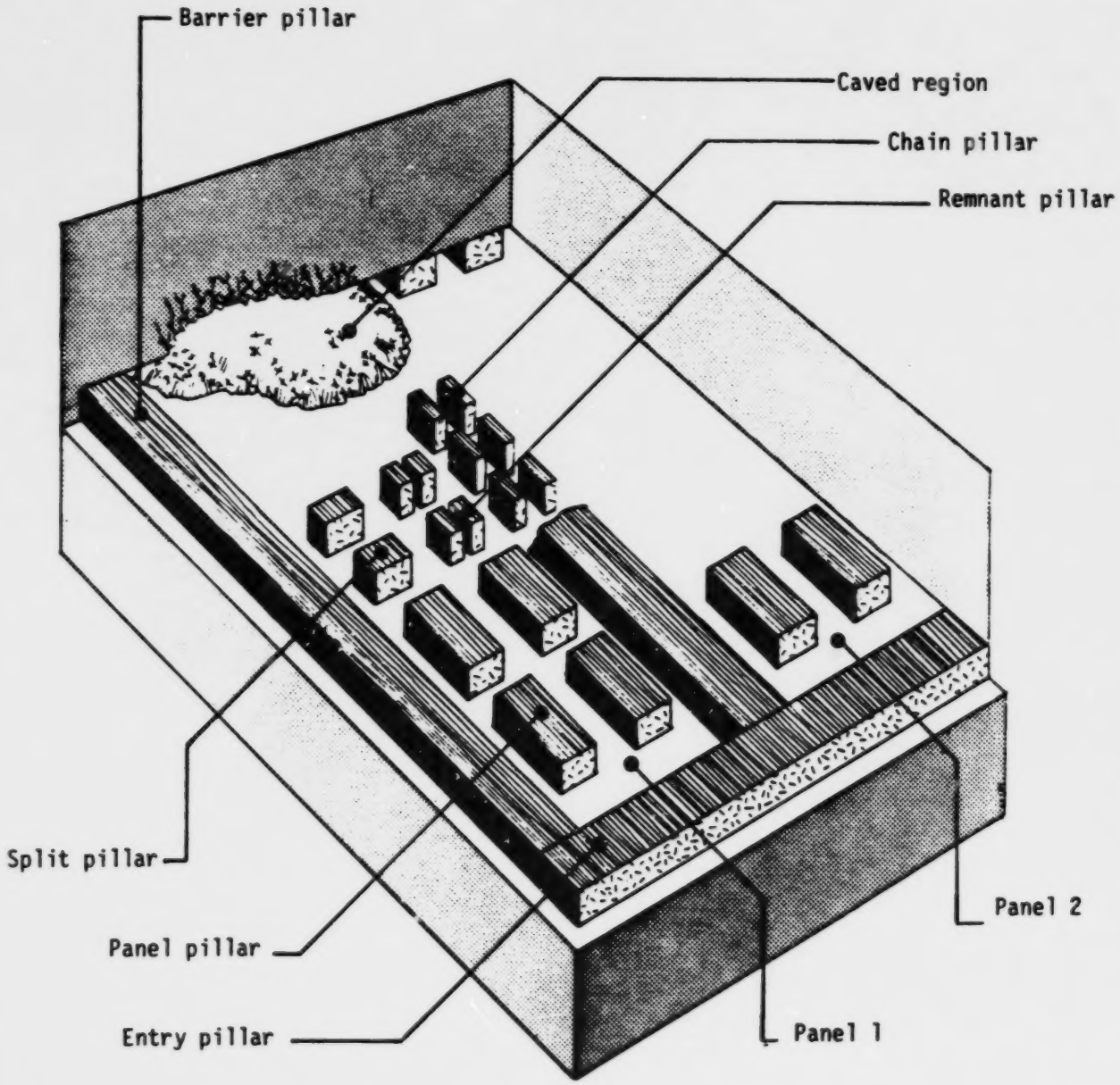

FIG. 1.2 COAL MINE PILLARS

PANEL PILLAR — PILIER DE PANNEAU
QUILLE

These temporary pillars are generally left at uniform intervals within a panel to provide roof support.

SPLIT PILLAR — PILIER REFENDU

A split pillar is a panel pillar which has been divided into two sections during recovery of ore from it, in order to enable production of a stope.

REMNANT PILLAR — PILIER RÉSIDUEL

A remnant pillar is defined as the remaining part of a split pillar. This type of pillar is completely extracted upon completion of mining but may collapse prior to this. This pillar does not actually serve as a support.

CHAIN PILLAR — PILIER EN CHAINE

A series of small pillars aligned in the direction of the separation axis between panels. They serve the same purpose as barrier pillars but enable a higher ore recovery rate and in some cases, displacement of the roof.

1.3 SPECIAL PILLARS

The third group of terms are used to designate pillars which serve a special purpose. Illustrated in figure 1.3, these pillars are defined according to the purpose they fill.

POST PILLAR OR YIELDING PILLAR

These terms originate from a mining method developed at the Strathcona mine. It defines a pillar specially designed to fracture and give in when the ratio of its length to its width reaches a predetermined value.

PERMANENT PILLAR OR TEMPORARY PILLAR

Permanent pillars ensure stability during the entire life of the mine and are not worked. Temporary pillars are recovered during mine production.

1.4 PILLARS OF THE MINES VISITED

The terms used at the mines visited are listed in table 1.1. Some of these terms have been defined in the preceding pages while others, such as major apex and longitudinal pillar, are defined and illustrated in the detailed mine reports presented in Annex 3.

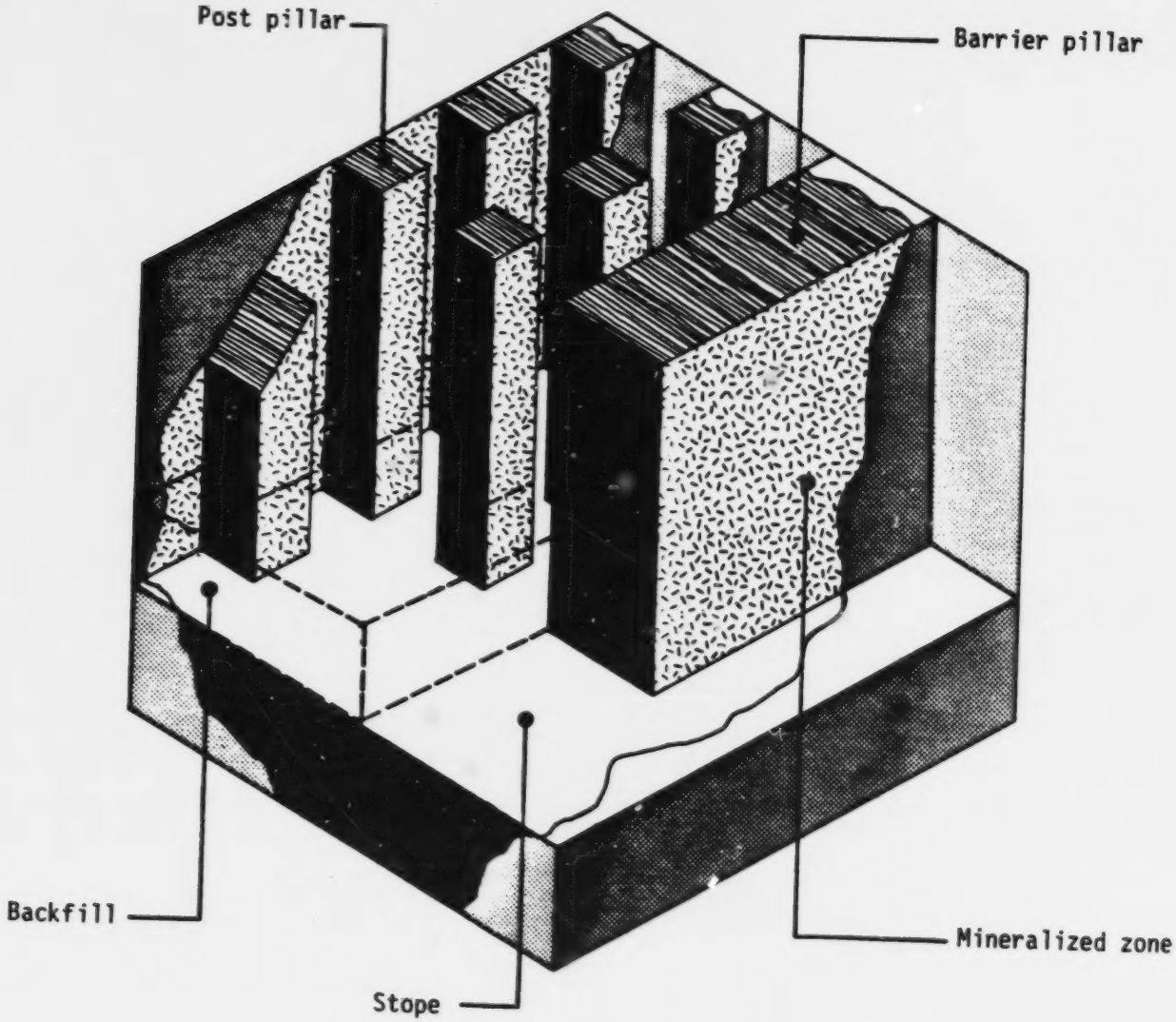

FIG. 1.3 SPECIAL PILLARS

TABLE 1.1: TERMINOLOGY OF MINE PILLARS

MINE NO. 1	MINE NO. 2	MINE NO. 3	MINE NO. 4	MINE NO. 5	MINE NO. 6
- Pilier de couronne ("crown piller")	- Pilier supérieur ("crown piller")	- Pilier permanent ("permanent piller")	- Pilier ("piller")	- "Rib piller", end piller", "transverse piller", "vertical piller"	- Pilier transversal
- Pilier principal ("major apex")	- Pilier inférieur ("sill piller")	- Pilier ("piller")	- Pilier de couronne ("crown piller")	- "Sill piller"	- Pilier longitudinal
- Pilier secondaire ("minor apex")	- Pilier de surface ("surface piller")	- Pilier de surface ("surface piller")	- Stot	- "Longitudinal piller"	- Pilier de surface
		- Pilier de couronne		- "Center piller"	
		- Stot		- "Surface piller"	

ANNEX 2

BIBLIOGRAPHIC RESEARCH

TABLE OF CONTENTS

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2.0 BIBLIOGRAPHIC RESEARCH

2.1 COLLECTION AND SELECTION OF WORKS

The plan of action adopted for obtaining documents will be outlined in this annex and the various works retained will be reviewed, underlining those which discuss pillar design methods.

During the course of our bibliographic research, we attempted to review all possible sources of documentation. First, we consulted several well known articles which deal with pillar design in order to become more familiar with the terminology used. We then retained key words in order to conduct a computer search.

The types of works available to us was discussed with the advisory committee, whose responsibility it was to inform us of other sources of information. We also consulted recent scientific journals (1982, 1983, 1984) in order to find articles not yet listed in computer banks.

. Types of works

The written sources retained during the course of our literature review are listed in table 2.1.

We were able to locate several sources of information, such as conferences and symposiums, periodicals and public reports, through computer searches.

TABLE 2.1 : SOURCES OF INFORMATION

COMPUTER DATA BANKS

. GEOREF, MINTEC, NTIS, COMPENDEX

CONFERENCES AND SYMPOSIUMS

. Australian and New Zealand Conference on Geomechanics
. International Conference on Stability in Underground Mining
. Congress of the International Society of Rock Mechanics
. Canadian Rock Mechanics Symposium
. U.S. Symposium on Rock Mechanics

PERIODICALS

. American Underground Space Association
. Bulletin de Liaison de Laboratoires des Ponts et Chaussées (France)
. C.I.M. Bulletin
. Engineering and Mining Journal (London)
. Geotechnics (England)
. Industrie Minérale - Les techniques (France)
. Institution of Mining and Metallurgy (London)
. International Journal of Rock Mechanics
. Canadian Journal of Earth Sciences
. Journal of American Society of Civil Engineering
. Mining Engineering (A.I.M.E.)
. Works of the Society of Mining Engineering (A.I.M.E.)
. Canadian Geotechnical Journal

PUBLIC AND INTERNAL REPORTS

. CANMET reports
. Reports presented by various commissions
. U.S. Bureau of Mines

. Selection of key words

The use of computer searches to assist in bibliographic research is useful only if relevant key words are retained. The key words retained in our study and an example of their use, are listed in table 2.2. Some of the key words retained were eliminated in order to avoid obtaining too great a number of articles (e.g. the word "mining" appeared in 49 041 titles). After having evaluated the number of titles obtained, we terminated our search at step no. 11 of table 2.2. For example, 287 references were obtained from the computer bank "GEOREF" alone.

These key words were subsequently used in three other computer searches. The data banks MINTEC, COMPENDEX and NTIS provided 667, 552 and 204 references respectively. Of course, some of the references were common to all of the banks. The references were this sorted to eliminate articles cited more than once.

. Additional references

The number of documents listed in data banks for the years 1982, 1983 and 1984 is probably incomplete due to the time required to bring the banks up to date. For this reason, we consulted the indexes of recent issues (1982, 1983, 1984) of the periodicals listed in table 2.1 and examined the programs of rock mechanics conferences held in 1983, 1983 and 1984.

TABLE 2.2 : ARTICLES OBTAINED FROM THE GEOREF DATA BANK

KEY WORDS	NUMBER OF ARTICLES
1. Pillar	161
2. Pillars	275
3. (1) or (2)	373
4. Mining	49 041
5. Mine	7 909
6. Mines	6 063
7. (4) or (5) or (6)	56 417
8. Underground (w) mining	144
9. Underground (w) mines	50
10. (8) or (9)	193
11. (3) and (7 or 10)	287

. Coding of documents

The titles and abstracts of the works were read, and those articles which were irrelevant or did not appear useful were eliminated from the initial list. The works which appearch on the final list were either obtained directly from resource persons or organizations or ordered from outside Quebec. In addition, some works were obtained through inter-library loans.

All works obtained were read and assigned a code based on the parameters listed in table 2.3. The codes were then compiled in a data bank set up especially for this research. The codification system presented in table 2.3 enables fast, easy identification of works related to a particular subject. The alphanumeric code reveals the region in question, the nature of the problem and technical data (e.g. design criteria) regarding each work.

The codes have been included in the list of references found at the end of this annex. The bibliography can therefore be used to assess the content of the large number of works retained. Of the more than 1 000 works evaluated during the course of the bibliographic research, approximately 830 were eliminated and some 170 retained based on this system. These articles come from more than 20 sources of information, as outlined in table 2.1.

TABLE 2.3 : CODING SCHEME

1. REGION
   a) Canada
   b) U.S.A.
   c) Europe
   d) U.K.
   e) Africa
   f) Australia
   g) South America
   h) U.S.S.R.
   i) Asia

2. GENERAL EVALUATION
   a) Very relevant
   b) Case history
   c) Theoretical
   d) Practical
   e) C and D

3. MATERIAL OVERLYING THE PILLAR
   a) Soil
   b) Lake, ocean, river or marshland
   c) A and B
   d) Nothing

4. FAILURE CRITERIA
   a) Surface pillars
   b) General pillars
   c) Tunnels

5. DESIGN
   a) Surface pillars
   b) General pillars

6. MONITORING
   a) Deformation/stress (pillar)
   b) Water pressure
   c) Deformation/stress (general)
   d) Overall
   e) Seismic impulses and microseisms
   f) E and D

7. STABILITY EVALUATION METHOD
   a) Empirical
   b) Analytical
   c) Numeric (finite element)
   d) Probabilistic
   e) Multiple
   f) In situ collection of data

8. LEGISLATION
   a) Laws
   b) Regulations - guidelines
   c) A and B

9. TYPE OF DEPOSIT
   a) Vertical pipelike magmatic (plug)
   b) Tabular $< 45°$
   c) Tabular $> 45°$
   d) Irregular

10. TYPE OF MINE
   a) Coal
   b) Salt
   c) Metal
   d) Multiple
   e) Other

11. SUPPORT
   a) Pillar

12. MINING METHOD
   a) Room-and-pillar
   b) Sublevel
   c) Block caving

13. SPECIFIC COMMENTS
   a) Rock classification
   b) Well-elaborated design criteria
   c) Future research subject
   d) Bibliography

14. REINFORCEMENT TECHNIQUES
   a) Anchor bolts
   b) Steel cables
   c) Chemical additives

Our computer data bank includes the following data:

- name of author(s)
- year of publication
- title of work
- source
- code

The bibliography included at the end of this annex constitutes a complete list of the works retained and includes a code indicating their principal characteristics. Based on this list, a systematic analysis of the literature was conducted, the objective of which was to more clearly define problems related to surface pillar design.

2.2 LITERATURE REVIEW

. Chronological classification of articles

The number of works retained is listed chronologically in figure 2.1. This figure illustrates years in which international congresses, national conferences and symposiums were held. The fourth congress of the International Society of Rock Mechanics held in Montreux, Switzerland in 1979 accounts to a large extent for the many articles published that year.

We would like to underline the growing interest expressed by many organizations engaged in the mining industry in rock mechanics since 1973. The development of this relatively young science and its increasing number of applications have contributed to the large number of works dealing specifically with mine pillars.

. Regional classification of articles

The place of publication of the articles retained is illustrated in figure 2.2. The United States and Canada rank first and second, accounting for approximately two-thirds of the total. Europe and Great Britain account for approximately one-sixth of the total and Africa, Australia, Asia and South America represent the remaining one-sixth. This distribution can be explained by the following factors:

YEARS	NUMBER OF ARTICLES
1965	-
1967	---
1968	---
1969	--
1970	----
1971	-----
1972	-----
1973	----------
1974	---------
1975	---------
1976	----------
1977	---------
1978	--------
1979	----------
1980	----------
1981	----------
1982	----
1983	----------

FIG. 2.1 CHRONOLOGICAL CLASSIFICATION

COUNTRY/CONTINENT	NUMBER OF ARTICLES
UNITED STATES	62
CANADA	47
EUROPE	30
AFRICA	9
AUSTRALIA	7
ASIA	7
SOUTH AMERICA	1

FIG. 2.2 REGIONAL CLASSIFICATION

- The majority of texts dealing with coal mines and written for the most part in Europe were not retained.
- There is a greater number of organizations and researchers in North America than elsewhere.
- There is a language barrier with respect to eastern European and Asian countries.
- Access to North American sources is readily available.

. Types of deposits

More than half of the works studied provide no information regarding the shape, dimensions and nature of the orebodies in question. Theoretical and/or practical generalities are discussed with no reference being made to particular types of deposits.

Tabular deposits dipping at less than 45° were the most frequently encountered in the works consulted. This is partially due to the large number of studies conducted on mines situated in deposits of sedimentary origin that have undergone very little change. Metal mines and coal mines were by for the types of deposits most often discussed. We should mention that we were particularly interested in metal mines.

. Details regarding articles on surface pillars

Eighteen articles made reference to or discussed surface pillar design. These articles can be classified as follows:

- ten articles related to metal mines: Page and Brennen, 1983; Tincelin, 1982; Commission of Inquiry - Belmoral mine, 1980; Touseull and Rich, 1980; Brissenden, 1980; Hedley et al., 1979; Sandy et al., 1977. 1976; Oko and Taylor, 1974; Southey, 1969; Tincelin and Riveline, 1965.

- five articles related to coal: Dejean and Arcamone, 1979; Babcock and Hooker, 1977; Wardell, 1976; Skelly and Loy, 1976; Orchard, 1969 and 1975.

- one article dealing with limestone: Bonvallet et al., 1978.

- one article discussing coal, metal, calcium, limestone mines etc: Maury, 1979.

The studies which made reference to surface pillars were often indirectly related to these pillars, conducted in response to particular ground conditions. For example, in cases in which open pit mines are subsequently mined underground, special studies on stress fields in surface are required, as in the case of the Shagani mine in Zimbabwe (Page and Brennen, 1983) and the Kidd Creek mine in Ontario (Hedley et al., 1979) where stresses around the surface pillar were evaluated by means of numerical methods. Some reports describing undersea mines (Southey, 1969, Orchard, 1969 and 1975) and several subsidence studies also discussed the behaviour

of surface pillars. Tincelin and Riveline (1965), for example, define a critical zone in which subsidence can create problems related to structure situated above underground mines. This zone is delineated by angles of influence, or hazardous zones, which are a function of the depth and dimensions of stopes under production. The nature of the overburden and rock mass, and drainage conditions are also cited as factors which influence the area of this critical zone.

Surface pillars in coal mines were the subject of numerous studies conducted in the United States. Guidelines and regulations have been developed for surface pillar design, particularly in mines near surface and underground bodies of water (Babcock and Hooker, 1977 Wardell, 1976; Skelly et Loy, 1976). These guidelines are generally conservative and provide minimum dimensions to be respected in the design of surface pillars. Nevertheless, it should be noted that these guidelines apply exclusively to coal mines and that, to date, no study has been conducted on surface pillars in metal mines.

Following the collapse of a surface pillar, inquiry commissions are sometimes formed to determine the causes and conditions of the event. For instance, the commission of inquiry investigating the Belmoral mine tragedy prepared a technical report providing details regarding the surface pillar. Maury (1979) and Dejean and Arcamone (1979) analyzed a number of collapses or cave-ins which have occurred in French mines and provided the dimensions of several surface pillars (crown pillars).

2.3 PILLAR DESIGN

Pillar design is based primarily on field data regarding the geology of the site and the mining method proposed. After the geometry of the underground openings required to carry out this mining method has been determined, criteria of failure for rocks and safety factors are taken into consideration.

. Safety factor

In the field of civil engineering the strength of materials can be evaluated with precision and it is relatively easy to preduct when failure of these materials will occur. This is due primarily to the quality and homogeneity of the materials used. Whether it is a question of tensile strength, compressive strength or shear strength, it is generally easy to determine safety factors to be used in the design of beams, concrete columns, etc. to prevent accidents resulting from unforeseen overloads (limited) or from a weakness or defect in the material.

For example, the average ultimate compressive strength " Rc" of a concrete column, as determined by standard tests on samples is compared to the load " p" which it must carry.

The ratio $\frac{Rc}{p}$ thus indicates the number of times the load carried by the concrete column can be multiplied before failure occurs. A factor of 1 indicates a state of equilibrium in which failure could occur at any moment. A safety factor of 1.2 ($\frac{Rc}{p} = 1.2$) is often used for construction materials. The dimensions of the material on which stresses will be acting are determined accordingly.

The strength Rc represents the in situ behaviour of the column and, in general, the following formula can be formulated:

$$Fs = \frac{Qu}{P}$$

where Fs = safety factor
Qu = ultimate in situ strength of material (In the above-mentioned, example, Qu≃Rc)
P = load applied to material

In the case of mine pillars, the approach to pillar design is different since their breaking strength cannot be determined with as much precision as can that of construction materials. As in civil engineering, the objective is to ensure that the load applied to the material, in this case a mine pillar, does not exceed its ultimate strength. However, it is difficult to determine the ultimate strength of the mine pillar due to a number of factors, such as joints, schistosity and heterogeneity of the rock, which influence the mechanical properties of the rock mass constituting the mine pillar. A number of researchers have attempted to approximate pillar strength by means of empirical formulas which take various parameters, such as the ratio of pillar width to pillar height, the volume of the pillar and its permeability, into consideration. Given the considerable differences between results of compressive strength tests conducted on small rock samples, researchers apply a safety factor to the laboratory strength value. This safety factor allows for differences between laboratory and in situ strength properties of rock and accounts for errors introduced by the various assumptions made in the design process. Duvall (1977) states the following safety factors have been used in the design of a large number underground openings:

$$\sigma_t \leq T_o / F_t$$

and $$/\sigma_c/ \leq C_o/F_c$$

where σ_t = maximum or critical tensile strength around opening

T_o = tensile strength of rock (laboratory)

F_t = safety factor for tension, 4 to 8

σ_c = maximum or critical compressive strength around opening

C_o = compressive strength of rock (laboratory)

F_c = safety factor for compression: 2 to 4

The larger safety values are recommended for long-term stability and for rocks having relatively weak joint systems, and the smaller safety factors are recommended for short-term stability and for rocks having no significant joint system (Duvall, 1977).

Goodman et al. (1980) propose the following formula for calculating the design strength (σ_p) of pillars:

$$\sigma_p = \frac{qu \cdot N(shape) \cdot N(size)}{F}$$

where qu = unconfined compressive strength of rock in pillars

N (shape) and N (size) = correction related to the shape and size of the pillar

F = safety factor for decreasing risks related to assumptions made in the design process.

Here again, we see that in order to determine the design strength of the pillar, a safety factor is introduced. The design strength (σ_p) could be used in the general equation $F_s = \frac{Qu}{p}$. We would obtain the following relationship which gives a safety factor (F_s) already weighted by the first factor (F):

$$F_s = \frac{qu \cdot N\,(shape) \cdot N\,(size)}{\frac{F}{p}}$$

Thus, in practice, a safety factor and an unknown factor are applied at two separate phases, the former being used in a general equation for calculating stability and the latter in the calculation of rock strength based on samples.

Three of the mines visited - no. 3, 4 and 6 - applied safety factors to strength values obtained in a laboratory. In the case of mine no. 6, reduction coefficients between 0.25 and 0.90 were used to weight the uniaxial compressive strength of the rock types encountered.

The design equations proposed in the articles reviews were primarily related to coal mines. Design methods for metal mines were virtually nonexistent.

. Design equations

To design pillars, it is necessary determine the strength of the pillar under various stress conditions. Criteria of failure and an equation representing the shear strength of the rock mass must be used. In the literature reviewed, the Mohr-Coulomb criteria of failure appears to be the most often used. It is expressed by the following formula:

where: $\tau = c' + \sigma' \tan \phi'$

$\sigma' = \sigma - u$

and : τ = shear strength

c' = cohesiveness

σ' = effective stress

σ = total stress

u = interstitial water pressure

This equation is that of a straight line whose slope equals " tan ϕ" and whose interception is c' (see figure 2.3).

Such a linear relationship is however, a conservative approximation . in several cases, it can be demonstrated that the relationship is not a straight line but a parabola. This is well illustrated in figure 2.4 taken from Hoek and Bray (1981).

The majority of authors who discussed pillar design use one criterian of failure based anly on stresses (Logie and Matheson, 1982). Consequently, the effect of thime is ignored. Page and Brennen (1983) also propose the use of deformation criteria to indicate when pillar failure will occur. Limiting themselves to criteria of failure based on stresses. Logie and Matheson (1982) outline a number of equations that may be used to evaluate pillar strenqth. These are described in table 2.4. To the equations mentioned by Logie and Matteson, we have added those encountered in the literature reviewed. In addition, additional safety factors are sometimes added to these equations.

The first equation with which pillar strength can be calculated is that presented by Duvall (1946) taken from Logie and Matheson (1982):

$$\sigma_p = \sigma_1 \left(A + \frac{BW}{H}\right)$$

$$\tau = c' + \sigma' \tan \phi'$$

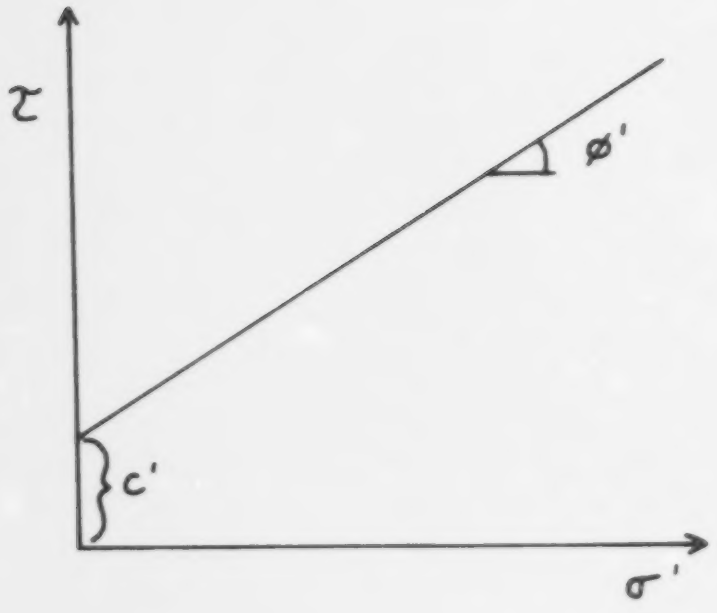


FIG. 2.3 MOHR-COULOMB FAILURE CRITERION

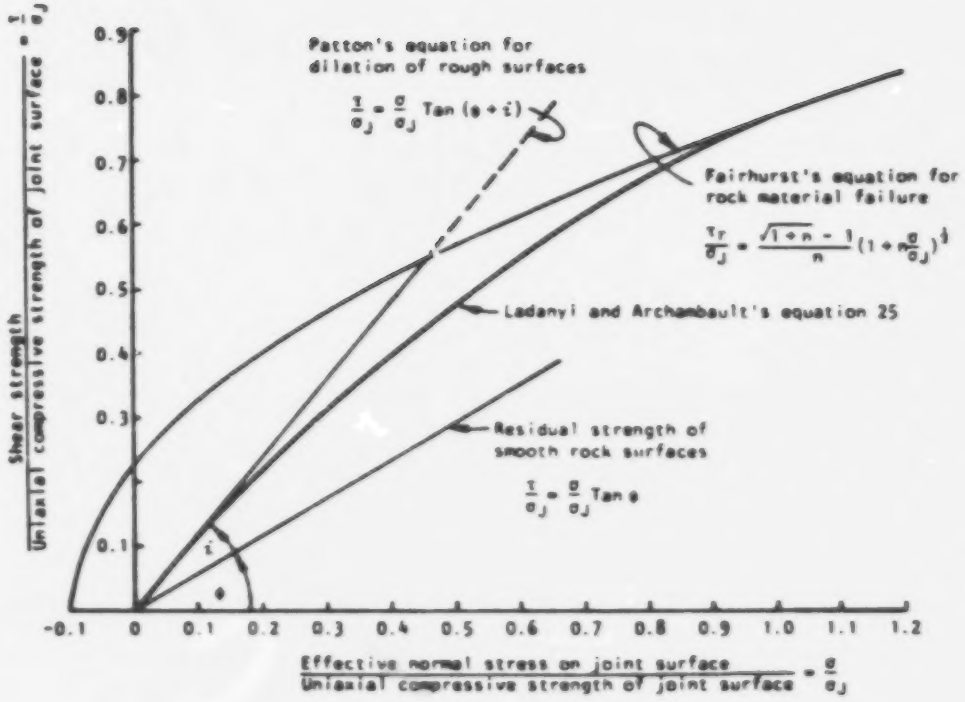


FIG. 2.4 TRANSITION FROM DILATION TO SHEARING PREDICTED BY LADANYI AND ARCHAMBAULT'S EQUATION

(Plotted for $i = 20^{o}$ and $\phi = 30^{o}$) (Hoek and Bray, 1981)

TABLE 2.4 : PILLAR DESIGN EQUATIONS

NO	NAME	EQUATIONS	REMARKS
1	Duvall (1946) Obert et Duvall (1967)	$\sigma_p = \sigma_1 (0.778 + 0.222 \frac{W}{H})$	- σ_p : pillar strength (psi) - σ_1 : uniaxial compresive strength of a cubical specimen - W,H : width and height of pillar - Safety factor of 2 to 4 - This equation relates to coal
2	Holland et Gaddy (1964)	$\sigma_p = k \frac{(W)}{H}^{0.5}$	- k : coefficient depending on coal - Widely used in North America - Safety factor of 1,8 to 2,2
3	Bieniawski (1967)	$\sigma_p = 1\,100 \frac{W}{H}^{0.16}$ for $\frac{W}{H} < 0.55$ $\sigma_p = 400 + 200 \frac{W}{H}$ for $\frac{W}{H} > 0.55$ $\frac{\sigma_p}{\sigma_c} = 0,645 + 0,355 \frac{W}{H}$ $\sigma_p = \left[0,64 + 0,36 \frac{W}{H}\right]^{\alpha}$	- For samples less than 60 inches on a side - For samples greater than 60 inches on a side - Normelizing equation using a 60 inch cube strength - σ_c : crushing strength (psi) - α : 1,4 for $\frac{W}{H} > 5$ - α : 1 for $\frac{W}{H} < 5$ - This equation relates to coal

TABLE 2.4 : PILLAR DESIGN EQUATIONS

NO	NAME	EQUATIONS	REMARKS
4	Salamon et Munro (1967)	$\sigma_p = \sigma_c \left[\frac{w^{0.46}}{H^{0.66}}\right]$ $\sigma_p = \sigma_s \dfrac{(Wp/Ws)^{0.46}}{(Hp/Hs)^{0.66}}$	- σ_c : uniaxial compressive strength of one foot cube of coal (psi) - Equation modified for samples of variable dimensions - σ_s : uniaxial compressive strength of a cubic sample - Wp : width of pillar - Hp : length of pillar - Hs : height of sample - used for oil shale
5	Wardell (1976)	$S = a \left[\frac{1}{H}\right] + b \left[\frac{W}{H}\right]^{0.5}$	- S : strength of pillar (psi) - a : constant (1000) - b : constant (20) - Factor of safety: 1.5 - used for coal
6	Agapito et Hardy (1982)	$\dfrac{\overline{\sigma v}}{\overline{\sigma o}} = 1 + B \left[\dfrac{\overline{\sigma h}}{\overline{\sigma o}}\right]^b$	- $\overline{\sigma v}$: pillar strength (psi) - $\overline{\sigma o}$: pillar strength at zero horizontal stress - $\overline{\sigma h}$: average horizontal pillar stress (psi) - B : constant of 3.2 - b : constant of 0.82

TABLE 2.4 : PILLAR DESIGN EQUATIONS

NO	NAME	EQUATIONS	REMARKS
6	Agapito et Hardy (1982) (suite)	$\overline{\sigma_c} = \sigma_1 \left[\frac{V}{V_1}\right]^{\alpha}$	- V volume of laboratory specimen - V_1 in situ volume of rock mass - α volume reduction coefficient
		$\overline{\sigma_p} = (\sigma_{h1}) F_1 + (\overline{\sigma_{vp}} - \sigma_{v1}) F_2$	- σ_{h1} initial premining horizontal stress - F_1, F_2 factors dependent on pillar shape - $\overline{\sigma_{vp}}$ average vertical stress at mid-pillar height - σ_{v1} initial or premining vertical stress - Used for oil shale
7	Hardy et Agapito (1975)	$\sigma_p = \sigma_c (Wp\ Wc)^{0.597} / (Hp\ Hc)^{0.351}$ $\sigma_p = \sigma_c\ 0.354 \left[\frac{W}{H}\right]^{0.597}$	- Used for oil shale
8	Hustrulid et Swanson (1981)	$\sigma_p = k\ p^{.50} \left[\frac{W}{H}\right]^{.50}$	- Used for coal - k : constant based on compressive strength of field or laboratory sample and side dimension of sample.

TABLE 2.4 : PILLAR DESIGN EQUATIONS

NO	NAME	EQUATIONS	REMARKS
9	Wilson (1972)	$\sigma_1 = \sigma_0 + \sigma_3 \tan \beta$ $\tan \beta = \frac{1 + \sin \phi}{1 - \sin \phi}$ $\frac{x}{h} = \left[\frac{1}{(\tan \beta)^{\frac{1}{\tan \beta}} (\tan \beta - 1)}\right] \cdot \ln \left[\frac{\sigma_v}{\sigma_0}\right]$	- σ_1 failure stress - σ_0 unconfined compression strength - σ_3 confining pressure - tan β triaxial stress coefficient - ϕ angle of internal friction - x depth of yield zone from the ribside - h maximum pillar stress - σ_v maximum pillar stress - Used for coal
10	Wilson (1977, 1981)	$r_b = r_0 \left[\left(\frac{(k-1)\sigma_0}{p^* (k+1)}\right)^{\frac{1}{k-1}} + 1\right]$	- r_b radius from centre of roadway - r_0 radis of the roadway - q stress field remote from excavation - k triaxial stress factor $\frac{1 + \sin \phi}{1 - \sin \beta}$ - Used for coal. Equation for calculating the depth of the yield zone into polar coordinates. - p strength of support and broken material

TABLE 2.4 : PILLAR DESIGN EQUATIONS

NO	NAME	EQUATIONS	REMARKS
11	Duncan et Devane (1983)	$\sigma_1 = \sigma_c + k\,\sigma_3$ $\sigma_1 = \frac{\sigma_c}{f} + k\,\sigma_3$	- σ_1 : compressive strength of rock material - σ_3 : confining pressure - σ_c : uniaxial compressive strength of intact rock sample - k : $\frac{1 + \sin\phi}{1 - \sin\phi}$ - f : rock mass disruption factor
12	Page et Brennen (1983)	$\frac{1}{\sigma_c} = 3.5 \left[\frac{\sigma_3}{\sigma_c}\right]^{0.75}$ (compression) $\frac{1}{\sigma_c} = 8 \left[\frac{\sigma_3}{\sigma_c}\right] + 1$ (tension) $\varepsilon = \frac{\sigma_3 - (\sigma_2 + \sigma_1)}{E}$	- ε : extension strain in the rock. A value of $500\mu\varepsilon$ was taken as being beyond failure. - E : modulus of elasticity (Young's)
13	Hoek et Brown (1980)	$\frac{\sigma_1}{\sigma_c} = \frac{\sigma_3}{\sigma_c} + \left[m\,\frac{\sigma_3}{\sigma_c} + s\right]^{1/2}$	- m and s: constants which depend on the properties of the rock

TABLE 2.4 : PILLAR DESIGN EQUATIONS

NO	NAME	EQUATIONS	REMARKS
14	Hedley et al. (1972)	$\sigma_p = \frac{Sv \cos^2\alpha + Sh \sin^2\alpha}{1 - R}$ $Qu + K \frac{w^a}{h^b}$ $F_s = \frac{Qu}{\sigma_p}$	- Sv : vertical stress - Sh : horizontal stress - α : dip of orebody - R : extraction ratio - Qu : pillar strength - w : pillar width - h : pillar height - K : strength of 1 ft. cube, psi - a and b : constants - F_s : safety factor
15	Goodman et al. (1980)	$\sigma_p = \frac{qu \cdot N\,(shape) \cdot N\,(size)}{F}$ $N\,(shape) = (0.875 + 0.250 \frac{W}{H})$ $N\,(size) = \left[\frac{h}{hi}\right]^{0,50}$	- qu : unconfined compresive strength - H : height of sample (unconfined pressure) - hi : minimum height of a cubical coal specimen - Used for coal

TABLE 2.4 : PILLAR DESIGN EQUATIONS

NO	NAME	EQUATIONS	REMARKS
16	Comité Français (1979)	$\sigma_c^L = \sigma_c^{-u} - Ks$	- σ^L : vertical stress in pillar - σ_c^{-u} : average ultimate strength - K : constant - s : typical difference of ultimate strengths
17	Yu et Toews (1976)	$F = \alpha I_1 + \sqrt{J_2} = k$ $I_1 = (\sigma_x+\sigma_y+\sigma_z) = (\sigma_1+\sigma_2+\sigma_3)$ $J_2 = \frac{1}{6}\left[(\sigma_x-\sigma_y)^2 + (\sigma_y-\sigma_z)^2 + (\sigma_z-\sigma_x)^2 + 6(\tau^2_{xy}+\tau^2_{yz}+\tau^2_{zx})\right]$ $\alpha = 2 \sin \phi / (\sqrt{3}\,(3-\sin \phi))$ $k = 6\,C \cos \phi (\sqrt{3}\,(3-\sin \phi))$	- I_1 : first invariant of stress tensor - J_2 : second invariant of stress deviation tensor - α,k : material constants

This equation is similar to that used by Goodman et al.(1980). However, Goodman et al. have replaced (A + BW) by corrections which take the shape and size of the pillar into account (table 2.4, equation 15). Bieniawski (1967) also proposes a similar equation but adds an exponent "α" which varies depending on the geometry of the pillar.

The equation which is most commonly used to calculate pillar strength is that employed by Holland (1964), Bieniawski (1967), Salamon and Monroe (1967), Hustrulid and Swanson (1981) (refer to Logie and Matheson) and Hedley (1972) and is expressed as:

$$\sigma_p = K \frac{W^a}{H^b}$$

where: σ_p : shear strength of pillar
K : coefficient which varies depending on ground type
W : width of pillar
H : height of pillar
a,b: constants determined by author

Some authors use normalizing equations based on the compressive strength (σ_c) of the rock mass (Bieniawski, 1967, and Agapito and Hardy, 1982 (refer to Logie and Matheson); Page and Brennen, 1983; Hoek and Brown, 1980). Most of these authors only consider two-dimensional stress distribution. Yu and Toews (1976), however, propose a numeric equation based on the finite-element method which takes three-dimensional stress distribution into consideration:

$$F = XI_1 + \sqrt{J_2} = K$$

The values I_1 represents the first invariant of stress tensor and J_2, the second invariant of stress deviation tensor in a three-dimensional model (table 2.4).

In determining the magnitude and direction of the in situ stress field in rock formations, some authors only take the static load of the rocks above the pillar into consideration. Hedley et al. (1972) suggest that normal stresses acting on discontinuities and the extraction rate also be taken into consideration and propose the following equation:

$$P = \frac{So}{I - R}$$

where So: normal stress acting on deposit, before mining
R : rate of ore extraction

For operations in inclined stopes, the value of So is modified as follows:

$$So + S_v \cos^2 \alpha + S_n \sin^2 \alpha$$

where α = dip of the orebody

The two equations give the following:

$$P = \frac{S_v \cos^2 \alpha + S_L \sin^2 \alpha}{1-R}$$

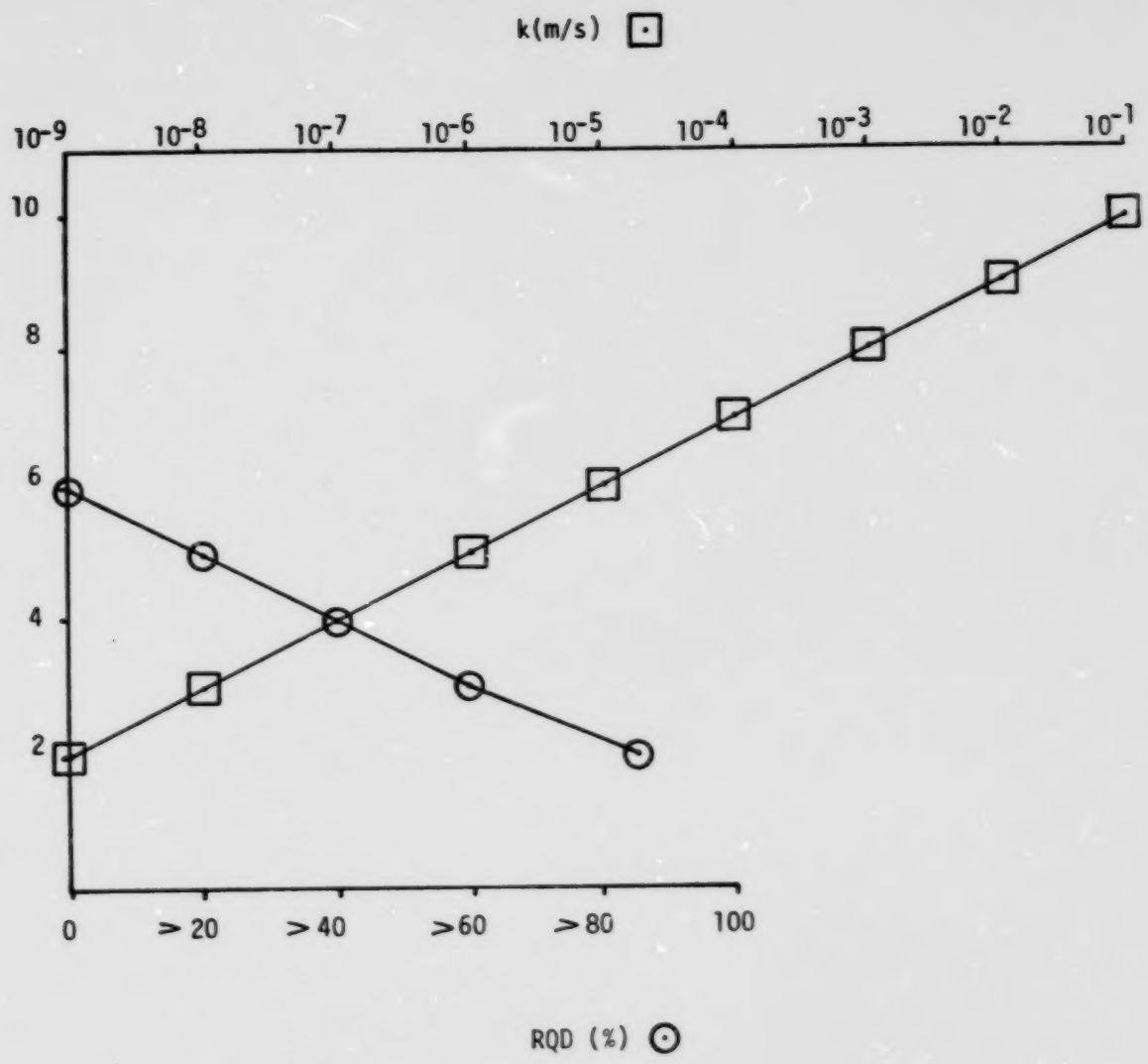
As can be seen, several approaches to pillar design are possible. In certain cases, emphasis is placed on the laboratory-obtained value for the compressive strength of a rock sample. A correction is then applied to the calculated value in order to obtain the true in situ value. Duncan and Devane (1983) propose a correction which can be calculated based on criteria such as the permeability and R.Q.D. of the rock mass (see figure 2.5).

Other authors who propose original approaches to pillar design are Lundquist and Heins (1972). They advance a probabilistic approach to the evaluation of changes in load. Page and Brennen (1983) add a criteria of failure to their equation based on strain:

$$\bar{\varepsilon} = \frac{\sigma_3 - (\sigma_2 + \sigma_1)}{E}$$

where $\bar{\varepsilon}$ = extension strain

E = Young's modulus of elasticity



$$\sigma_1 = \frac{\sigma_c}{f} + K\sigma_3$$

$$\sigma_{mo} = \frac{\sigma_c}{f}$$

FIG. 2.5 EVALUATION OF FACTOR "f", DUNCAN AND DEVANE (1982)

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ANNEX 3

MINE VISITS

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3.0 MINE VISITS

3.1 SELECTION OF MINES

The mine sites visited were selected by the advisory committee. Selection criteria were based on characteristics of mining properties which are representative of the present situation in underground mines.

In order to guide the committee in its choice, the study team prepared brief descriptive outlines of nine mines, underlining the local geology, the nature of the surface pillar(s), surface conditions and the methods used for ground control.

Six mines were selected, five of which are located in Quebec, and one in Ontario. The committee decided, for reasons of confidentiality, that the names and locationsof the mines should not appear in the report. The mines visited are therefore identified by number (from 1 to 6) and described in annex 3. Each case history is presented is detail following the format outlined in table 3.1.

Certain elements are described in greater detail for some mines than for others as a result of the priority given to these elements by the management of the mine and of the amount of documentation available at each mine. The representative of Roche Limited gave particular attention to surface pillar design criteria and stability control methods.

TABLE 3.1 : FORMAT OF MINE REPORTS

1.0 TYPE OF DEPOSIT - MINING METHOD

- 1.1 BRIEF HISTORY OF THE MINE
- 1.2 MINING METHOD
- 1.3 TERMINOLOGY EMPLOYED
- 1.4 SURFACE PILLAR DESIGN CRITERIA
  - 1.4.1 Study of water conditions
  - 1.4.2 Soils overlying the mineralized zone
  - 1.4.3 Model of the mine
  - 1.4.4 Rock mechanics
  - 1.4.5 Support
  - 1.4.6 Instrumentation and control
  - 1.4.7 Geometry of surface pillars
  - 1.4.8 Rock mass evaluation
  - 1.4.9 GENERAL
- 1.5 ASSESSMENT OF THE VISIT

3.2 MEETINGS WITH MINE MANAGEMENT

Meetings were he'd with the key personnel of each mine, which included moduled the general manager and engineers and geologists responsible for underground works, to acquire information regarding their approach to surface pillar design. The subjects discussed in these meetings were those outlined in a questionnaire prepared to orient the discussions. These included the views of management with regard to overall mine stability, surface pillar design criteria and stresses and various in situ and laboratory works. Personnal view regarding the subjects were also expressed at these meetings.

Two days were generally required at each mine for the meetings, discussions and visit to the relevant underground mine locations.

3.3. SYNTHESIS OF THE MINE VISITS

Table 4.1, taken from the main text, has been included here to clearly illustrate the works conducted at each of the mine sites visited. The seven types of activity will be discussed separately in order to underline the most positive elements of each category. As previously explained, the symbols represent the following:

- ○ little detail
- ◑ moderately detailed studies
- ● very detailed studies
- ▬ no study realized/not applicable

TABLE 4.1 SUMMARY OF STUDIES AND ANALYSES REGARDING SURFACE PILLARS

STUDY	MINE NUMBER					
	1	2	3	4	5	6
HYDROGEOLOGY						
. Visual evaluation	○	-	-	-	○	○
. Specific analysis	-	◑	◑	●	-	-
SURFACE CONDITION						
. Soil removal	-	-	◑	-	-	-
. Drainage	-	◑	◑	●	-	-
. Drill holes	-	◑	◑	●	-	○
. Laboratory analyses	◑	◑	○	●	-	-
. Drawings and plans	○	◑	◑	●	○	○
. Seismic surveys	-	-	-	○	-	◑
MODELLING, SIMULATION						
. Computer model	-	-	◑	◑	-	◑
. Simulation	-	-	-	-	-	-
ROCK MECHANICS						
. Internal program	◑	◑	◑	●	◑	◑
. Outside studies	-	○	◑	○	-	◑
. Laboratory testing	-	-	○	○	-	-
SURFACE PILLAR SUPPORT TECHNIQUES						
. Conventional bolting	-	◑	◑	-	◑	◑
. Grouted rebars	-	◑	●	-	-	-
. Resin-anchored rebars	-	-	-	-	-	●
. Loose fill	-	-	-	-	-	-
. Hydraulic fill	-	●	-	-	-	-
. Consolidated fill	-	-	◑	-	-	●
. Grouted cables	-	-	-	-	-	●
. Shotcrete	-	◑	-	-	-	-
INSTRUMENTATION, STABILITY CONTROL						
. Measurement of displacements	○	-	-	●	-	●
. In situ measurement of stress	-	-	-	-	-	○
. Visual inspection	○	◑	○	○	○	○
. Systematic monitoring	◑	●	●	-	-	-
ROCK MASS EVALUATION						
. Integrated classification	●	○	○	◑	-	○
. Type of rock	-	◑	-	-	-	-
. Structure	-	-	○	◑	◑	○
. R.Q.D.	-	-	○	-	◑	●

3.3.1 Hydrogeology

WORKS	MINES					
	No.1	No. 2	No. 3	No. 4	No. 5	No. 6
Visual evaluation	○	-	-	-	○	○
Specific analysis	-	◑	◑	●	-	-

Neither surface nor ground water pose major problems to three of the six mines visited. This was determined by conducting visual inspections of the stopes. At these mines, a traditional pumping system combined with a drainage network is sufficient to control underground water. The joint networks prevent any substantial accumulation of water. Consequently, no further investigations of water conditions have been conducted by these mines.

The operators of mines no. 2 and no. 3, where water is pumped at a rate of 80 to 100 L/sec (1 050 to 1 320 gpm), has attempted to determine the groundwater level and the surface conditions which accelerate or hinder the flow rate in order to adapt the underground pumping system to the particular conditions of the site.

Only the operators of mine no. 4, recognizing that the presence of water posed serious problems to the production and stability of the mine (initial flow rate of 108 L/sec of pumped water), pursued a detailed hydrogeological analysis. The objective of this analysis was to modify existing conditions or, as mentioned in one of their studies, to depressurize and dewater the 40 metres of overburden and the rock mass in the upper part of the mine so that the groundwater level would

drop to an elevation of -180 m. In order to do this, the management of the mine determined the precise properties of the ground, installed a elaborate drainage network and strictly monitored changes caused by the pumping system.

3.3.2 Surface conditions

WORKS	MINES No. 1	No. 2	No. 3	No. 4	No. 5	No. 6
Soil removal	-	-	◑	-	-	-
Drainage	-	◑	◑	●	-	-
Drill holes	-	◑	◑	●	-	○
Laboratory analysis	◑	◑	○	●	-	-
Plans and drawings	○	◑	◑	●	○	○
Seismic survey	-	-	-	○	-	◑

This table outlines the works conducted by the six mining companies is order to delineate and evaluate the overburden overlying the upper stopes.

The radical measure consisting of removing the overburden above the stopes to be mined to the surface was taken at only one mine for a small number of stopes. Nevertheless, we know that this method is used elsewhere in the province of Quebec. At the mine visited this decision was made for economic reasons based on the projected revenue resulting from such an operation. In three of the mines visited, the measure would not have been profitable because of the large quantity

of overburden. The two remaining mines could not undertake such a measure due to the presence of a lake over their stopes.

The drainage of stopes is generally a routine operation in most mines. Drainage often occurs naturally, i.e. joint systems are often sufficiently developed to allow the flow of water and thus prevent the formation of high water pressure within the rock mass. Mine operators need only channel the water to decantation basins and pumping stations. Three of the mines visited took direct measures to accelerate the drainage of the overburden. Mine no. 4 is particularly interesting since various methods were combined to dewater the overburden.

The analysis of holes drilled from the surface is one of the most commonly used methods to evaluate the thickness of the overburden and, at the same time, the profile of the underlying bedrock. This method is used almost universally. Often, the borehole cores are obtained from initial exploration drilling carried out to discover or delineate the mineralization. At mines no. 3 and no. 4 however, special drilling programs were implemented in order to increase the accuracy of initial estimates. These programs are often necessary since little exploration drilling is done in the upper mine levels.

The mines which carried out drilling programs to determine the thickness of the overburden also took samples and had the necessary analyses conducted to identify the geotechnical properties of the various layers encountered. These data are used either in the development of drainage programs or in the evaluation of the effect of an accidental surface pillar failure, as was the case at mine no. 4.

Plans, longitudinal sections and cross sections of the surface profile are always available whereas the profile of the bedrock is genearlly less defined due to the above mentioned reason. However, it may not always be necessary to determine the exact location of the overburden/bedrock contact (for instance, if the deposit ends 5 or 10 metres from the surface). We would simply like to state that it is always difficult to determine the location of the bedrock and consequently, to clearly indicate it on mining plans.

Seismic surveys were conducted at only one of the mines visited where it was necessary to determine the thickness of overburden under five metres of water.

3.3.3 Modelling, simulation

WORKS	MINES No. 1	No. 2	No. 3	No. 4	No. 5	No. 6
Computer model	-	-	◑	◑	-	◑
Simulation	-	-	-	-	-	-

Three of the six mines visited used computer models taken from special studies to evaluate the overall stability of their stopes. This technique was applied specifically to surface pillars is only two cases.

At all of the mines visited, the operators showed a considerable degree of reticence in discussing the content and specific applications of these studies. Due to the following reasons, we feel that it is normal that the mines are reluctant to publish this information:

1- the application of analysis models based on the finite-element method is relatively new in the field of underground mines in Quebec;

2- these studies are based on analysis conducted at a specific point in the life of the mine under fixed conditions which can change quickly;

3- a relatively small amount of imput data are used in the model;

4- the results of such analysis are general and, in the majority of cases, open to interpretation.

Since the operators regard computer models as one of several decision-making tools, they fear that the results will be miscontrued to such a point that they will be interpreted as a reflection of the only possible reality. Nevertheless, we feel that there should be no reluctance in wishing to describe as accurately as possible a reality as complex as a rock mass.

We have obtained some information on the subject, primarily regarding mines no. 3 and no. 4. Having visited these mines, we clearly feel that the results of such models cannot be and are not considered as the only elements to be taken into account in surface pillar design.

3.3.4 Rock mechanics

WORKS	MINES No. 1	No. 2	No. 3	No. 4	No. 5	No. 6
Interval program	◑	◑	◑	●	◑	◑
Outside studies	-	○	◑	○	-	◑
Laboratory testing	-	-	○	○	-	-

All of the mines visited expressed a certain willingness to have an active rock mechanics department. The idea of entrusting the responsibility of rock mechanics programs to one or more spesialists is generally well accepted. Although not all of the mines devote large amounts of time and money to rock mechanics programs, they all conduct a minimum number of works.

In most cases, the rock mechanics teams work in collaboration with the geology and production departments. Their role is to gather data regarding rock mass properties and required support methods and to make recommendations regarding the type of mining method to be used. The rock mechanics teams generally retain the services of outside laboratories to obtain information regarding the physical properties of the rock masses under study.

In more complex cases, in which mathematical tools are required, mine directors can retain specialized consultants or consulting firms in this field. At mines no. 3, no. 4 and no. 6, outside specialists were retained to evaluate mine stability and make recommendations for surface pillar design.

3.3.5 Surface pillar support

WORKS	MINES No. 1	No. 2	No. 3	No. 4	No. 5	No. 6
Conventional bolting	-	◑	◑	-	◑	◑
Grounted rebars	-	◑	●	-	-	-
Resin-anchored rebars	-	-	-	-	-	●
Loose fill	-	-	-	-	-	-
Hydraulic fill	-	●	-	-	-	-
Consolidated fill	-	-	◑	-	-	●
Grouted cables	-	-	-	-	-	●
Shotcrete	-	◑	-	-	-	-

We could write at great length regarding the support techniques used by these mines, all of which expressed considerable interest in this aspect of mining. However, the applications of support systems for surface pillars alone are much more limited.

In old stopes, conventional bolting in the roof and walls is frequently encountered. In mines no. 2 and no. 3, several old surface pillars required additional bolting due to deterioration of the rock. Grouted rebars were used, a technique which will likely become standard practice since this type of bolting does not require maintenance. The stability of these pillars is also ensured by backfilling underlying stopes, a technique which tends to limit deformation. Mine no. 6 uses a combination of resin-anchored rebars, grouted steel cables and consolidated backfill to support the hanging wall constituting the surface pillar.

It is difficult to assess whether the support techniques employed by these mines are over or under-estimated. Nevertheless, the fact remains that the operators recognize the importance of ensuring the long-term stability of surface pillars.

3.3.6 Instrumentation and control

WORKS	MINES No. 1	No. 2	No. 3	No. 4	No. 5	No. 6
Measurement of displacements	○	-	-	●	-	●
Measurement of in situ stresses	-	-	-	-	-	○
Visual inspection	○	◑	○	○	○	○
Systematic monitoring	◑	●	●	-	-	-

The objective of using measuring instruments is to monitor in situ rock mass behaviour. It is a means of acquiring information and of verifying design hypotheses.

Mines no. 4 and no. 6 use rod extensometers to detect displacements of the rock mass constituting the surface pillar. The objective of these two mines in using these instruments is to gather data on ground behaviour as the mining operations progress towards the upper part of the orebody. In fact, the surface pillars have not yet actually been formed; the two mines are currently working on the final design (thickness) of these pillars. Instruments were installed to provide data to be used in pillar design if the results obtained do not correspond to initial predictions. The importance of the use of this type of equipment is widely acknowledged by the operators of these two mines.

Natural stresses acting on the stopes have not been measured at any of the mines visited. At mine, no. 6, however, the consulting team retained to analyze the overall stability of the mine by means of a numerical model performed some such measurements. The results obtained by the use of the " overcoring" method show that the ratio of the horizontal stress to the vertical stress (K_o) varies considerably along the two principal axes of the orebody. In the upper part of the deposit, K_o equals approximately 3.5 along the longitudinal axis and between 1.0 and 2.0 along the transvese axis. We point this out in order to underline the necessity of recognizing this factor before proceeding to computer-assisted design, as has mine no. 6. Mine no. 4 has used average K_o values, compiled by other mines whose orebodies occur in similar formations. It intends, however, to conduct several analyses of this type regarding its own rock mass.

All of the mines visited depend to a large extent on their personnel in the evaluation of stability. We were told that all underground employees are capable of detecting signs of instability and of evaluating the overall competence of the rock mass. The inspection of the stopes is carried out by all levels of supervision, from the foremen to the manager of the mine, including the rock mechanics engineers. This is certainly an excellent practice but one which should be given greater attention. It is always possible that a sector of the mine be overlooked or forgotten or that information be lost in the shuffle. We have obtained details of two systems which have been adopted to avoid such possibilities.

First, the management of mine no. 2 has included a special section in the foremen's daily log for the systematic evaluation of openings. In the space entitled " ROCK STABILITY" , the foreman evaluates the condition of the rock mass on a scale of 1 to 6 and reports situations which may be cause for concern such as the falling of rocks or the presence of excessive water. This evaluation is conducted throughout the mine.

Second, no. 2 and no. 3 have developed specific inspection standards for surface pillars. The standards are defined in advance, the program team is identified and the frequency of the inspections is determined. A form has been prepared which clearly lists the points to be verified and all data gathered, as well as any necessary corrective measures taken, are recorded. We feel that this truly systematic procedure is very effective. A similar procedure is used at mine no. 1.

It should be noted that although these systematic inspections were started only a few years ago (1980 - 1981), those persons involved state that they recognize to a greater and greater extent the value of such a technique.

3.3.7 Rock mass evaluation

WORKS	MINES No. 1	No. 2	No. 3	No. 4	No. 5	No. 6
Integrated classification	●	○	○	◑	-	○
Type of rock	-	◑	-	-	-	-
Structure	-	-	○	◑	◑	○
R.Q.D.	-	-	○	-	◑	●

This table draws one's attention to the notion of " knowledge" of the rock mass within which mine workings have been developed. In our opinion, this is the first step of any serious analysis. Of the various methods that have been developed to evaluate rock quality, integrated classification techniques are designed to combine the greatest number of parameters which may contribute to a realistic description of the rock mass. From Terzaghi (1946) to Deere (1964) and Laubscher and Taylor (1977), to name only the most well know authors, the common objective of all authors has been to define the rock mass as objectively as possible. In order to illustrate the evolution of these classification, we have presented table 3.1[1], taken from the Master's thesis of P. Legast (1981). The table clearly shows that the most recent classifications attach considerable importance to the role played by discontinuities in the stability of rock masses. A whole set of recommendations will be made based on this theory, regarding the self-supporting capacity of the rock mass considered, support techniques, etc.

(1) Table presented in the main text.

Only one of the mines visited systematically uses Laubscher's classification a variant of that of Bieniawski (CSIR), originally developed for tunnel digging works. Mine no. 4 uses this classification together with that of the Norvegian Geotechnical Institute (NGI) developped by Barton et al. for the evaluation of surface pillar. At mines no. 2, no. 3 and no. 6, the two methods are used to a small extent in restricted areas generally as a base for the development of numerical models.

The case of mine no. 2 is interesting in that a number of years ago, the geology department of this mine determined the existence of nine rock classes at its property. The mine operators were accustomed to having a weighted evaluation of the rock mass on the geological plans. This evaluation was not conducted in as rigorous a fashion as the classifications described above but its objective was to provide the management of the mine with a coherent guide regarding the quality of the rock mass. Its only shortcoming, however, is the subjective manner in which ratings are assigned to the rock mass. In fact, the geologists of this mine analyze exploration borehole cores based on the type of rock encountered. For example, the rock may be a massive sericite which corresponds to type 3-4, an altered sericite which corresponds to type 4-6 or a meta-anorthosite which corresponds to type 0-1. Each of these rock types has a distinct behaviour and requires special attention. Therefore, the operators have a working tool which can be used to evaluate in advance ground conditions in which they intend to develop new stopes or galeries.

The geology department of mine no.2 gathers data regarding major geological structures for the preparation of plans. This procedure is well-established and is undertaken prior to the development of each stope. In general, however, we noticed that less information in gathered regarding the surface pillars than regarding other areas of the mine. This can be explained by the fact that mine operators are looking for mineralizations above anything else. However, the fact remains that it is necessary to acquire extensive knowledge regarding the rock mass in which the surface pillar will be formed.

Mine no. 6 in particular has recognized the significant role of discontinuities in stope stability. In order to do so, it determines the rock quality designation (R.Q.D.) based on borehole cores, a technique which it has been using for more than five years. We were told by the management of this mine that a site having a high R.Q.D. value, and therefore, possibly competent, was selected for the installation of a crushing chamber. In several cases, the evaluation of the R.Q.D. can be sufficiently accurate for the needs of the mine.

3.4 MINE VISIT REPORTS

As previously mentioned, a specific format was followed in the preparation of the reports on the mine visits. Given the considerable amount of information gathered at each mine visited, each mine report has a table of contents.

The following sections give more details.

MINE NO. 1

ASBESTOS MINE: BLOCK CAVING

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1.0 ASBESTOS MINE: BLOCK CAVING

1.1 BRIEF HISTORY OF THE MINE

The first tons of ore were extracted in 1878, at which time the mine operated as an open pit. During the initial years of operation, 330 000 metric tons of ore were mined annually. In 1951, the bottom reached the 165.3 m mark above sea level and, since then, the mine has operated underground.

The first level was set at + 114 m and is now exhausted. Since 1978, the second level, which is situated at + 1 m, produces 1 100 00 metric tons of ore annually.

Two companies each own a part of the deposit. The company whose property we visited is currently mining a section of approximately 282 m X 165 m, with an average dip of 40°.

The footwall of this massif-type deposit is divided by a major shear zone of 10 to 12 metres in thickness. The hanging wall is located on the neighbouring property.

The host rock is composed of partially serpentinized peridotite. The geological rock formations are heavily jointed and have a wide range of competence. Mineralization has occurred throughout the complex joint network.

1.2 MINING METHOD

The method of mining currently employed is block caving. It should immediately be noted that surface pillars no longer exist at this mine but were destroyed years ago by block caving. The advantage of this method rests in the fact that it can be used to evaluate and modify the natural conditions of stability of the rock mass.

The operations of this underground mine have been simplified and are illustrated in figure 1.1. In block caving, the first step consists in inducing the caving of a section between + 113.7 m and the bottom of the former shaft located at + 168.5 m in order to extract a minimal thickness of approximately 50 m.

This zone is referred to as the first lift. Subsequently, a second level is prepared in order to extract ore from the zone between + 1 m and + 113.7 m, the second lift. In block caving, mining progresses towards the bottom of the deposit in slices of approximately 100 m in thickness.

Over the last four years, the LHD extraction system (LOAD-HAUL-DUMP) has been used to recover ore from the drawpoints. Loaders on wheels which have access to the drawpoints of the mining levels are used to haul the ore.

The number of drawpoints is determined by the volume of ore situated above the zone of influence and by the total production required. Two important factors must be considered in the use of this method. First, the conditions favouring the caving of the rock mass in such a way that ore can be advantageously recovered from the drawpoints must be determined. Secondly, the permanence of these structures and of the

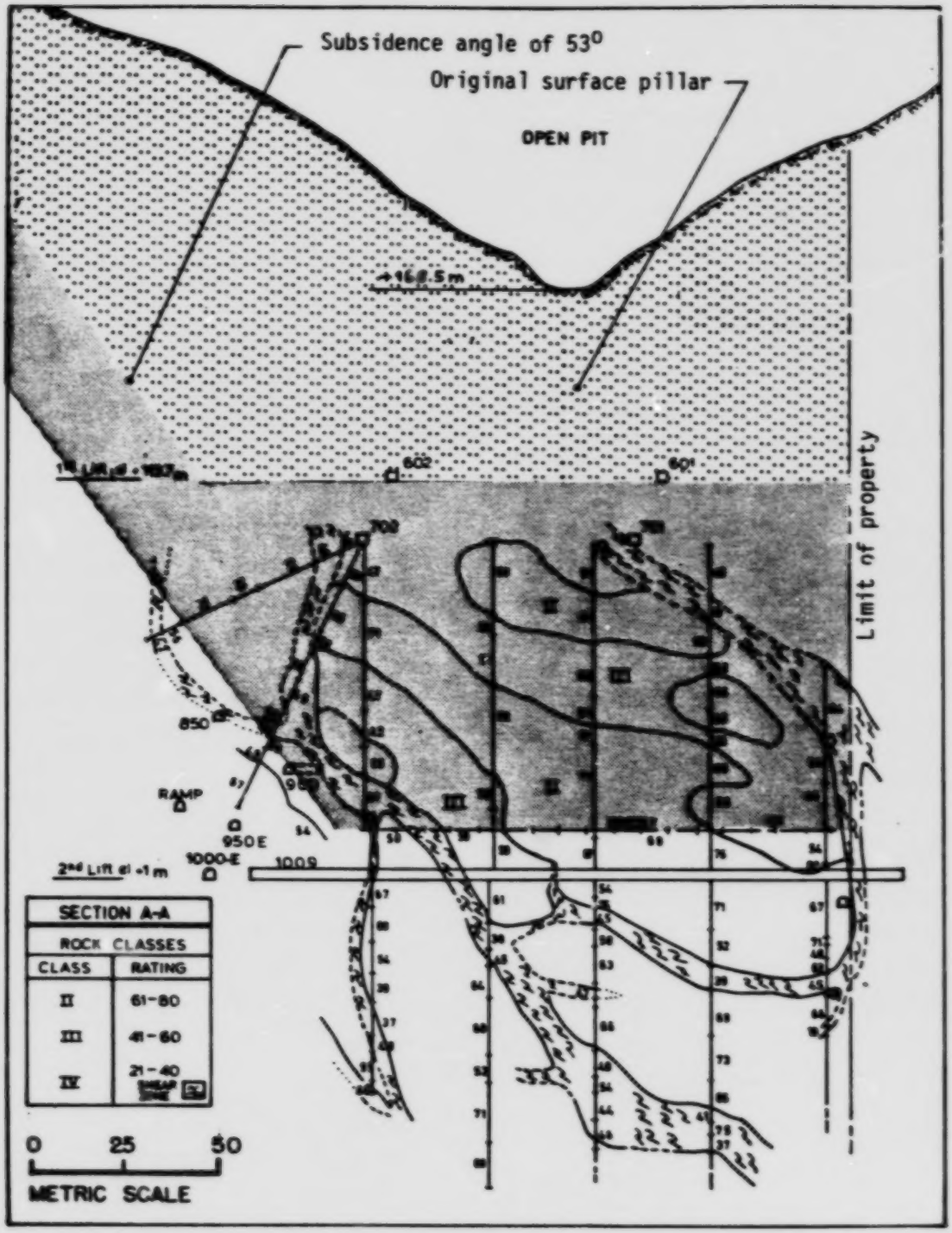


FIG. 1.1 MINING METHOD

drifts which link the drawpoints together and to other services (e.g. raises, garage) must be ensured.

This method of mining depends on factors which, when combined, cause the division into blocks and subsequent loss of stability of the rock mass.

1.3 TERMINOLOGY USED

During the course of our bibliographic research, we compiled a list of the terminology used in this abestos mine. In block caving, pillars are not required except at the level of the cones and extraction drifts situated beneath each block. Consequently, only two terms refering to pillars are used:

- crown pillars
- major and minor apexes (principal and secondary pillars)

These types of pillars are illustrated in figures 1.2 and 1.3, taken from a report published by this company. It is interesting to note that figure 1.2 represents an explanation of the probable origin of the term crown pillar. The shape of the protective rock mass cut above the haulage drifts following the grizzly method resembles a crown.

In the LHD technique, the pillars or apexes, as illustrated in 1.3, ensure the stability of the haulage drifts and drawpoints. Although this terminology will not be retained for the purposes of our study, it may serve to provide a greater understanding of the case history of this mine.

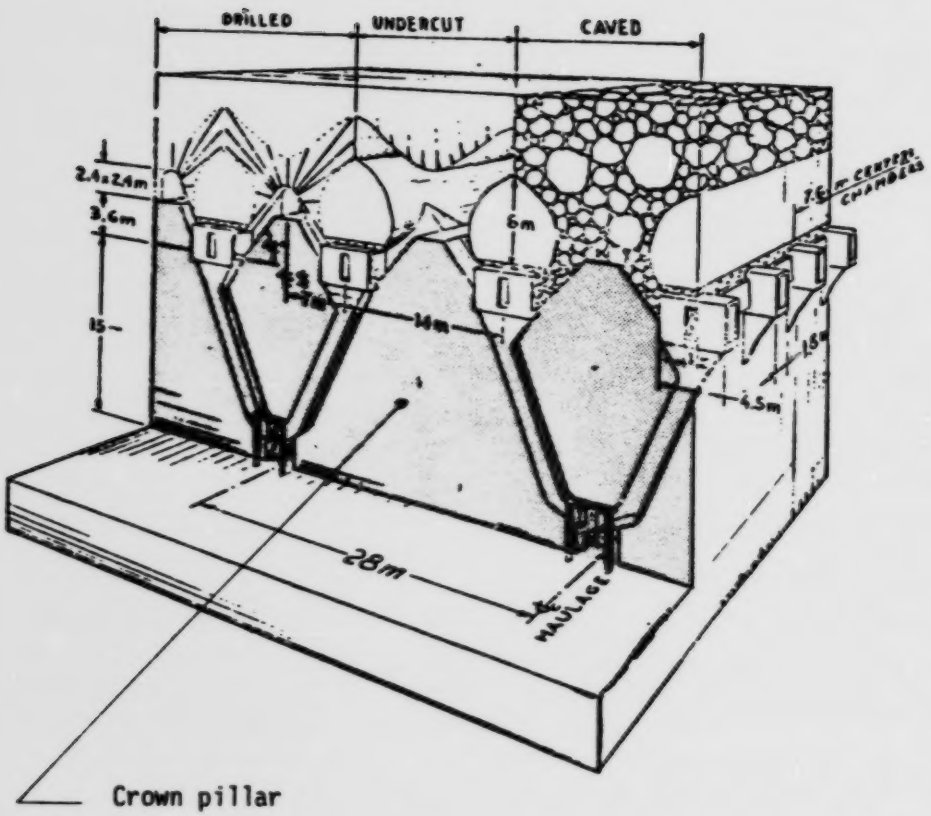
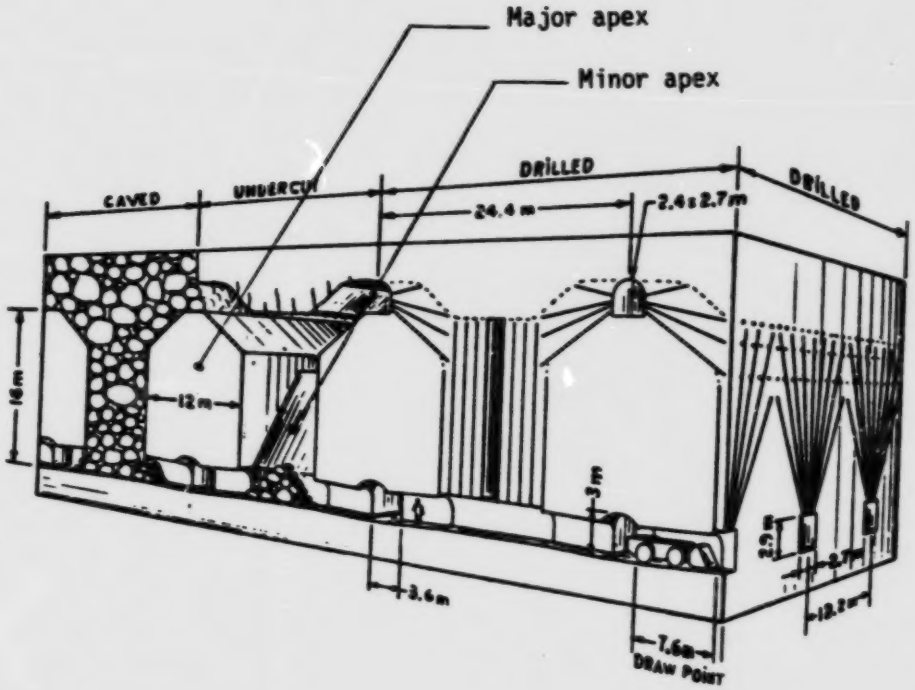


FIG. 1.2 GRIZZLY METHOD



CAVING STEPS AND DESIGN DIMENSIONS OF THE LHD METHOD

FIG. 1.3 MAJOR AND MINOR APEXES

1.4 SURFACE PILLAR DESIGN CRITERIA

Although the use of block caving fails to provide information on surface pillars, this mining method introduces us to a rock mass evaluation technique which is inexpensive and very useful to the operators. Within the framework of this study, it would be very useful to know which natural characteristics have the greatest effect on the stability of the rock mass, how to act on these characteristics in order to prevent the caving of the rock mass or in other words, to maintain its stability, and what surface effects are caused by the caving of a large rock mass.

The engineering team of this mine has asked itself these same questions. Consequently the need for an effective rock mass evaluation technique by which a maximum amount of data could be processed became apparent. The primary objective of this technique is to assist in the design of block caving systems and service openings. This technique will be described in further detail in section 1.4.8.

1.4.1 Study of surface and ground water conditions

The permeability of the fractured surface pillar (caved-in blocks) is high enough to prevent the development of water pressure in the pillar. No water infiltration problem has been detected by the management of the mine and no local hydrogeological study has been conducted.

1.4.2 Soils overlying the mineralized zone

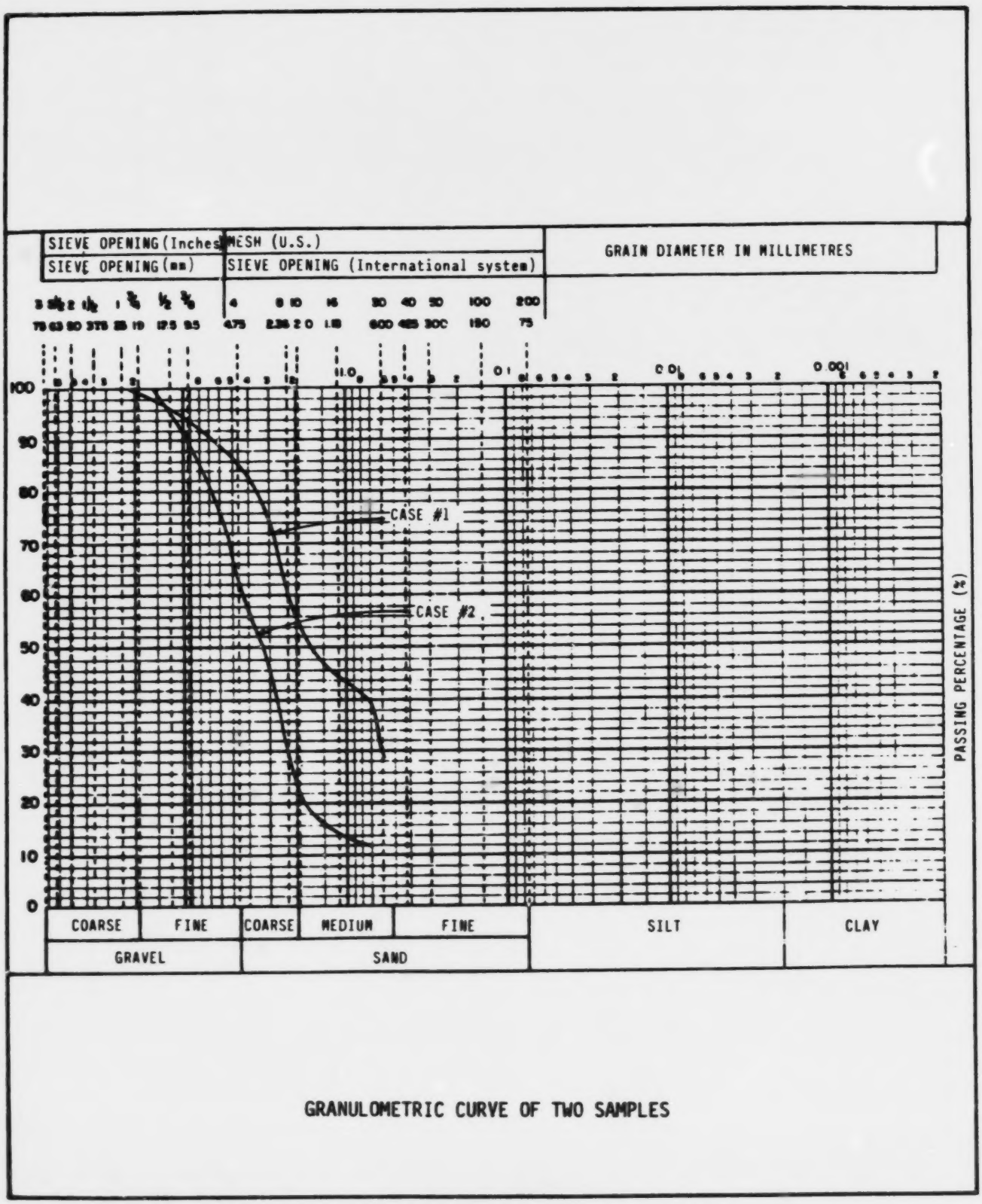
In order to avoid problems of ore dilution by the overburden, it was completely removed from the region of influence of the mine. The thickness of the overburden was approximately 9 m. For the last few years, mill tailings have be deposited in the zone of influence above the stoping area. The management of the mine has indicated that the presence of these tailings does not present major problems, with the exception that the extraction of ore from underlying stopes must be carried out in a well-determined sequence in order to prevent the formation of chimnies into which tailings could enter, dropping suddenly in large quantities into the drawpoints. The potential risk of liquefaction of the tailings in the fractured rock mass is virtually nonexistent. The coarse grain size (figure 1.4) of the tailings and rock mass renders the area highly permeable and the possibility of the build-up of substantial water pressure in the rock mass is minimal, if not nonexistent.

1.4.3 Model of the mine

No model has been used at time mine.

1.4.4 Rock mechanics

The rock mechanics program adopted by the management is based on Laubscher's rock mass classification (1977) which is described in further detail in section 1.4.8. The analysis of borehole cores and the mapping of drifts are the most important data in the application of this program. In addition, information gathered by the miners can also be of considerable significance.



GRANULOMETRIC CURVE OF TWO SAMPLES

FIG. 1.4 GRANULOMETRY OF MINE TAILINGS

Miners involved in surveillance programs (rock mechanics, safety, etc.) can contribute valuable information which can be used in the design and/or modification of mining plans. For example, the detection of water or unusual noise by a driller should be taken into account by the management of the mine in its rock mechanics program. In addition to the program, systematic investigations are carried out monthly to verify the stability of the stopes.

1.4.5 Support

During the early phases of mine development and ore extraction, a passive support system was employed. Square concrete structures were used in the drifts above the grizzlies and the principal draw-points were reinforced with H beams called steel sets. For the last few years, however, active support systems, such as anchored rock bolts, grouted cables and a combination of rock bolts, wire mesh and shotcrete have been employed.

The management decided to adopt an active support system because of the predominant role played by the rock mass structure in the overall stability of openings. Over the last few years, the rock machanics program of this company has been centred principally on the description of borehole cores and the mapping of all accessible drifts and has led to practical, profitable and efficient results.

The preparation of mining plans is based primarily on the rating obtained by the classification. The application of the classification in the design of mining plans is illustrated in figures 1.1, 1.5 and 1.6.

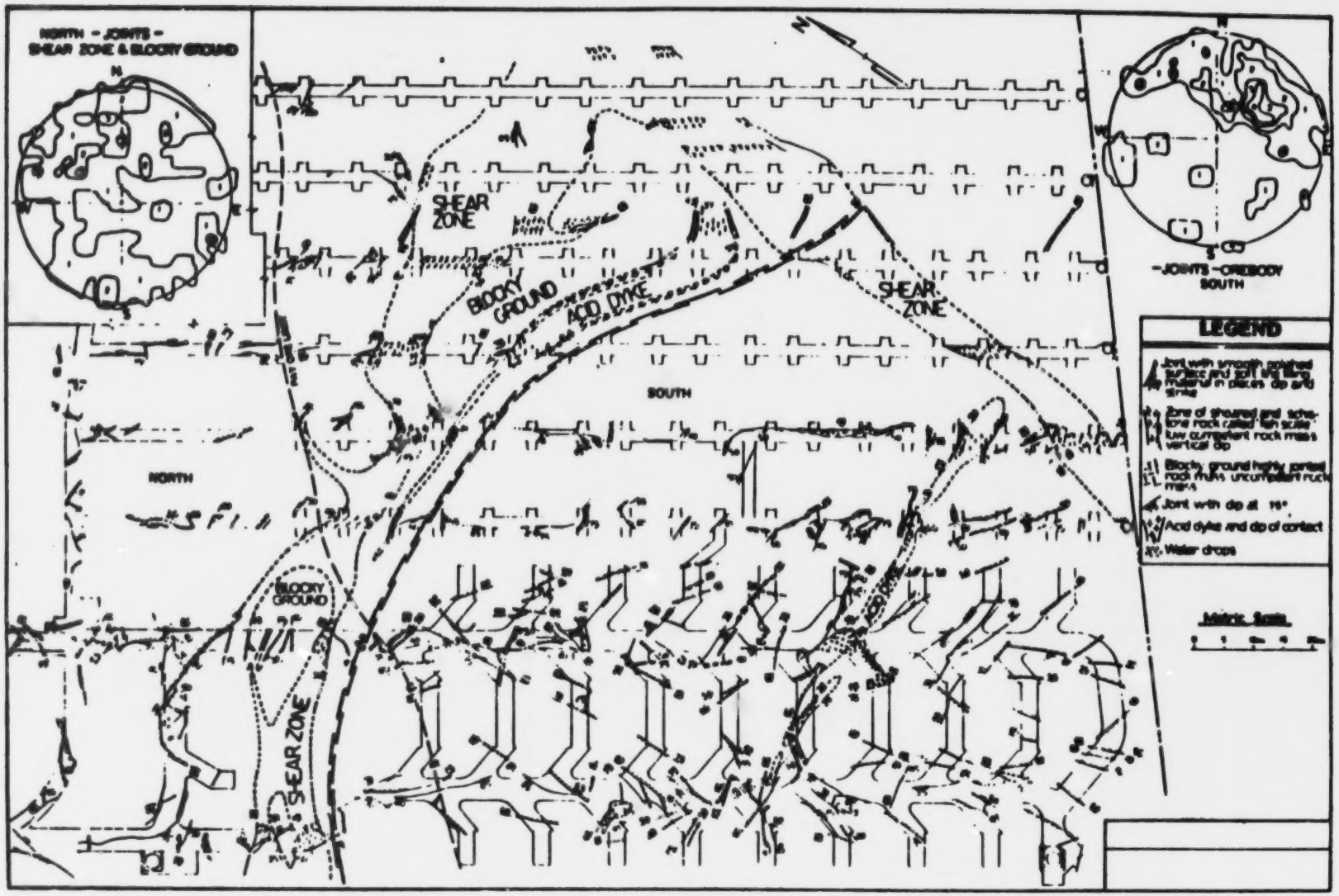


FIG. 1.5 STRUCTURAL GEOLOGY AND STATISTICAL STUDY OF JOINTS FOR A GIVEN LEVEL

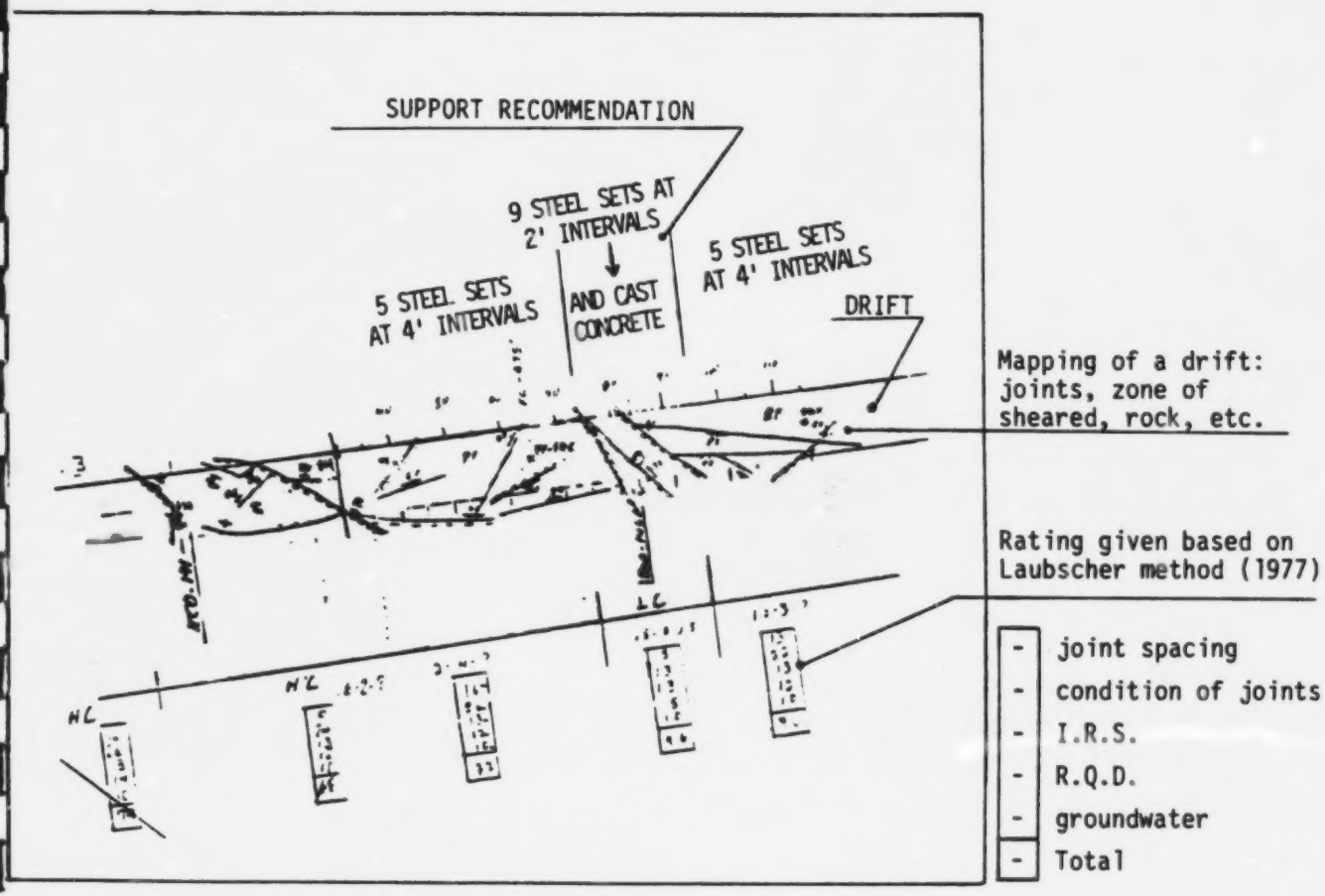

SUPPORT RECOMMENDATION
9 STEEL SETS AT 2' INTERVALS
AND CAST CONCRETE
5 STEEL SETS AT 4' INTERVALS
5 STEEL SETS AT 4' INTERVALS
DRIFT
Mapping of a drift: joints, zone of sheared, rock, etc.
Rating given based on Laubscher method (1977)
- joint spacing
- condition of joints
- I.R.S.
- R.Q.D.
- groundwater
- Total

FIG. 1.6 EXAMPLE OF THE USE OF THE LAUBSCHER METHOD IN A DRIFT

Figure 1.1 represents a cross section of a mineral deposits and illustrates the different rock classes found in it. Figure 1.5 shows a detailed plan view of one level; structural geology and statistical studies of joints are used for adjusting mining plans and for determining weak zones that will be crossed by drifts. Figure 1.6 illustrates an example of a weak zone (rock mass rating of 46) which has been given additional support. The active support system adopted by the management enables both better support cost control and the use of the most appropriate quantity and type supports for each particular situation.

1.4.6 Instrumentation and control

A surveillance program of subsidence breaks has enabled the precise determination of the location of the zone of influence above the mine. Surface subsidence measurements and readings of extensometers installed at certain depths have provided a subsidence angle of 53° with the horizontal. The location of the visible limit of subsidence at the surface is obtained by conducting annual surveys using standard theodolites.

The group with which we met seemed quite satisfied with the instruments used. An extensometer anchored at three points was installed in a hole drilled from the surface, the longitudinal axis of which forms a minimum angle with the subsidence plan. This enables the measurement of the rock mass displacement and the determination of the horizontal movement of the subsidence zone.

The team has been using this instrument for close to four years and has encountered no major problems. With regard to underground openings, a company policy requires the head of the rock mechanics department to conduct monthly evaluations of the quality of the drifts. The objectives of these inspections is to verify support conditions, to evaluate the effect of mining operations on the openings and to detect any sign of change which may be cause for concern in the hope that problems can be corrected from the first signs of instability.

1.4.7 Geometry of surface pillars

Given that the only surface pillar was block caved, the management has turned its attention to the design and surveillance of principal and secondary pillars overlying the stopes and haulage drifts. Since the geometric aspect is no longer a crucial parameter, the management is more concerned with controlling subsidence.

1.4.8 Rock mass evaluation

The rock mass evaluation is based on the classification scheme proposed by Laubscher (1977). Under this classification, the following parameters are taken into consideration:

1. R.Q.D. (rock quality designation)
2. I.R.S. (intact rock strength)
3. Joint spacing
4. Condition of joints
5. Groundwater

It should be noted that this classification is based on a technique developed by Bieniaswki (1974). Each of the five parameters chosen by Laubscher has its own value scale; a rating system based on criteria predetermined by the author is used to access the rock mass (table 1.1, figure 1.7, table 1.2).

An example of a rating assigned to a rock sample from a borehole core is illustrated in figure 1.8. The sum of the points assigned to each of the parameters determines the class to which the rock belongs. Laubscher's classification is divided into five classes, with value ratings of 20 per class. The classification has, therefore, a rating range from 0 to 100 (table 1.1).

A rock assigned a rating of 92 is a very competent, class 1A rock. Laubscher has also determined that certain factors may be used for adjusting the rating of the rock mass. These factors include weathering, field and induced stresses, changes in stress, strike and dip orientation and blasting effects.

Adjustments enable operators to take the particular conditions encountered during mining operations into account. However, it must be noted that these adjustments or reduction percentages of the initial rating assigned to the rock must be calculated by persons having extensive knowledge of underground extraction stopes. The possible reduction percentages of the ratings assigned to four of the five parameters mentioned above are shown in table 1.3. The total adjustment is the product of the specific adjustment factors of each of the conditions found in the rock mass studied.

TABLE 1.1

ROCK MASS CLASSIFICATION (LAUBSCHER, 1977)

Table 1 Classification of variations in jointed rock masses

Class	1		2		3		4		5	
Rating	100–81		80–61		60–41		40–21		20–0	
Description	Very good		Good		Fair		Poor		Very poor	
Sub-classes	A	B	A	B	A	B	A	B	A	B
Item										
1 RQD, %	100–91	90–76	75–66	65–56	55–46	45–36	35–26	25–16	15–6	5–0
Rating	20	18	15	13	11	9	7	5	3	0
2 IRS, MPa	141–136	135–126	125–111	110–96	95–81	80–66	65–51	50–36	35–21	20–6 5–0
Rating	10	9	8	7	6	5	4	3	2	1 0
3 Joint spacing	Refer to Table 2									
Rating	30									0
4 Condition of joint	45° ..... Basic angle of friction ..... 8°									
	Refer to Table 3									
Rating	30									0
	Inflow per 10-m length			0	25 l/min		25–125 l/min		125 l/min	
	OR									
5 Groundwater	Joint water pressure / Major principal stress			0	0.0–0.2		0.2–0.5		0.5	
	OR									
	General description		Completely dry		Moist only		Moderate pressure		Severe problems	
Rating	10				7		4		0	

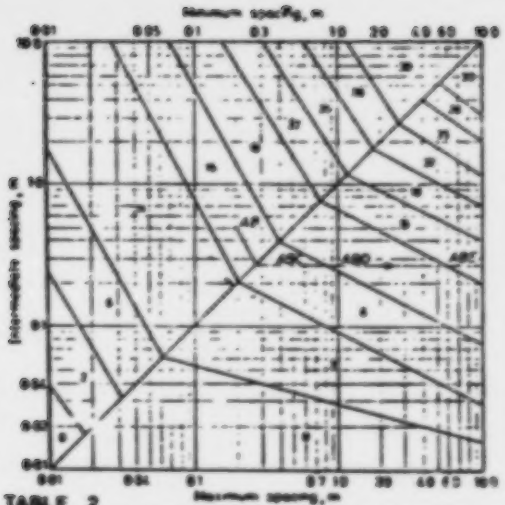


TABLE 2

Ratings for multi-joint systems: example. Joint spacing *A*, 0.2 m; *B*, 0.5 m; *C*, 0.6 m; *D*, 1.0 m; *E*, 7.0 m; *AB*, 15; *ABC*, 6; *ABD*, 11; *ABE*, 15

FIG. 1.7 JOINT SPACING RATING (LAUBSCHER, 1977)

TABLE 1.2

ASSESSMENT OF JOINT CONDITIONS

(LAUBSCHER, 1977)

Table 3 *Assessments of joint conditions* (adjustments as combined percentages of total possible rating of 30)

Parameter	Description	Percentage adjustment
(A) Joint [illegible] (Large-scale)	[illegible] unidirectional	[illegible]
		[illegible]
	Curved	[illegible]
		[illegible]
	Straight	70
		70
(B) Joint surface (Small-scale)	Serrated	[illegible]
		[illegible]
	Smooth	[illegible]
		[illegible]
	Polished	[illegible]
		[illegible]
(C) Alteration zone	Softer than wall rock	[illegible]
		70
(D) Joint filling	Coarse hard-sheared	[illegible]
		[illegible]
	Fine hard-sheared	[illegible]
		[illegible]
	Coarse soft-sheared	70
		70
	Fine soft-sheared	[illegible]
		[illegible]
	Gouge thickness	[illegible]
	< Irregularities	35
	Gouge thickness	20
	> Irregularities	12
	Flowing material	11
	> Irregularities	0

TABLE 1.3: ADJUSTMENT OF RATINGS (LAUBSCHER, 1977)

TOTAL POSSIBLE REDUCTIONS, %

	RQD	IRS	JOINT SPACING	CONDITION OF JOINTS	TOTAL
Weathering	95	96		82	75
Field and induced stresses				120-76	120-76
Changes in stress				120-60	120-60
Strike and dip orientation			70		70
Blasting	93			86	80

With this procedure, the classification scheme can be applied uniformly, that is, the basic evaluation is the same for all cases studied (all the mines). The adjustments enable the specific characteristics of a rock mass, mining method, etc. to be taken into consideration.

For the specific case which interests us, the series of steps to be taken to evaluate the rock mass is illustrated in figure 1.9. Data obtained from borehole cores, the mapping of drifts and, in certain cases, from rock mechanics laboratories are compiled in a computer program. Through the use of a computer program, considerable time can be saved in the determination and localization of rock classes and errors can be avoided. The evaluation of borehole cores or drift maps can be conducted by an engineer, geologist or technician. However, two to three months of training are required before the user can apply the identification technique with competence and efficiency.

At mines in which underground works are extensive (development of numerous drifts), it is preferable that at least two people gather the data required to establish the classification of the rock mass. A trained person can cover approximately 80 m of drifts per day. Shear zones are carefully noted for they can cause major weaknesses in underground workings.

With the classification scheme adopted by the management of the mine, it is possible to verify whether the data obtained from the borehole cores are similar to those observed in situ. In general, ratings obtained from borehole cores are similar to those obtained in the drifts. However, in the mapping of the drifts, particular attention must be given to the distinction between natural joints in the rock

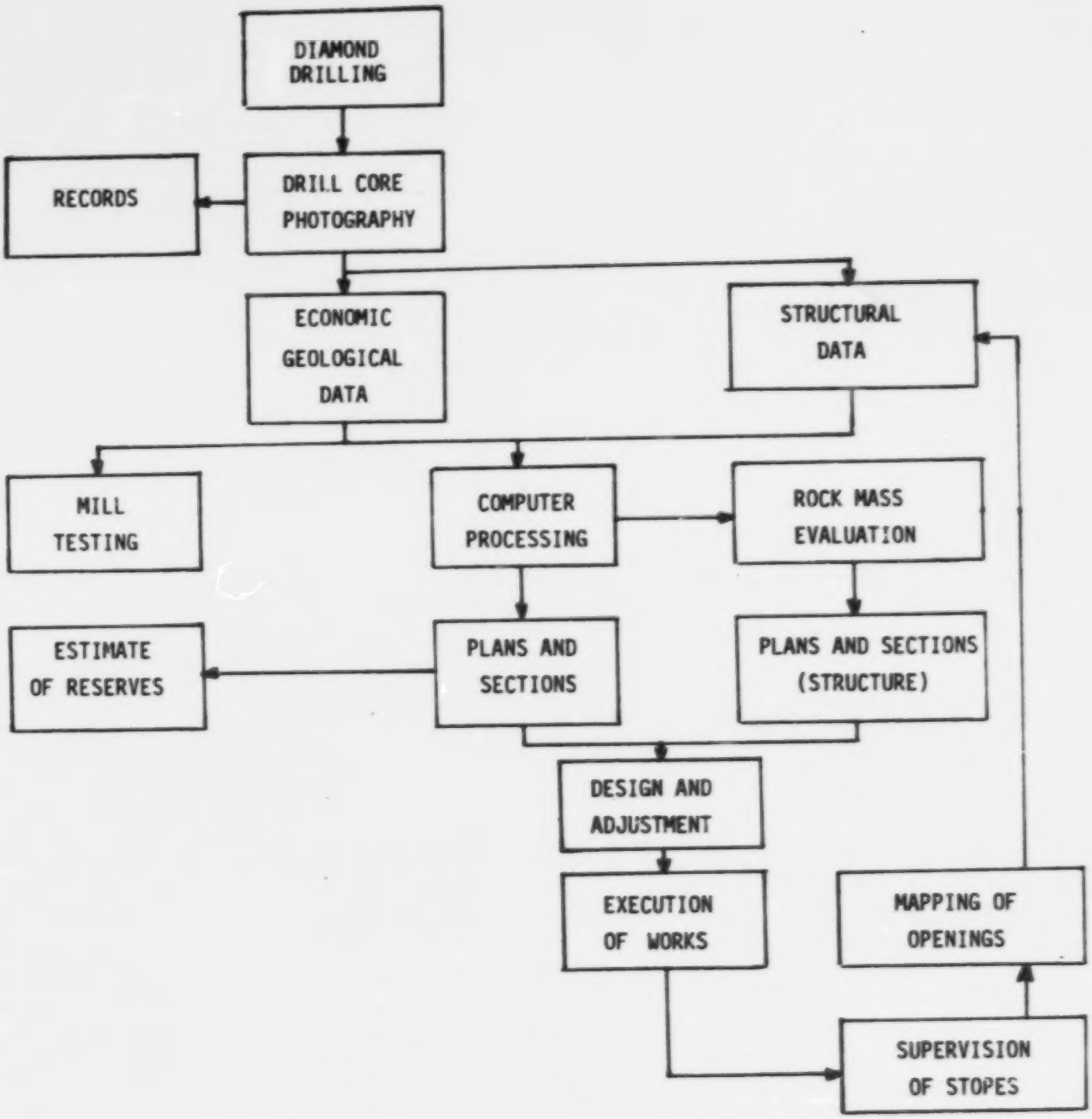


FIG. 1.9 METHODOLOGY EMPLOYED BY THE MANAGEMENT OF THE MINE TO CONDUCT ROCK MASS EVALUATIONS

and joints which may have been created by the blasting of the opening. The latter must be excluded from the evaluation.

The close correlation between the ratings of borehole cores and *in situ* ratings was obtained using a 33 m X 33 m diamond drilling pattern. The average length of the holes was approximately 100 m.

Of the parameters used in the classification, the uniaxial compressive strength of intact rock can be determined in a laboratory. Figure 1.10 illustrates an example of a uniaxial compressive strength test in which the rock sample fractured at 33 800 psi (238 MPa). It should be noted, however, that the management of the mine does not insist that such tests be conducted and that values obtained from load point tests are also satisfactory.

1.4.9 GENERAL

No further information exists regarding the mine which is relevant to this study.

1.5 ASSESSMENT OF THE VISIT

This case study provides a detailed description of the approach adopted by the management of the mine to deal with stability problems encountered in underground stopes. This approach is based primarily on the evaluation of the rock mass. The management has been able to apply a rock mass classification technique, in this case, that of Laubscher, in a very simple and economical manner in order to determine the type of support required to ensure the stability of the drifts, drawpoints and other necessary openings. Furthermore, through the use of their approach it is

File: 3400 Date: 11/30/82

STRESS-STRAIN RELATIONSHIPS OF ROCK - ASTM D 3148

Ident.: Ra: 975

Sample #: 1 Description:

Specimen #: B
Diameter, In. 1.24
Area, Sq.In. 1.21
Length, In. 2.80
Date Tested: 11/28/82

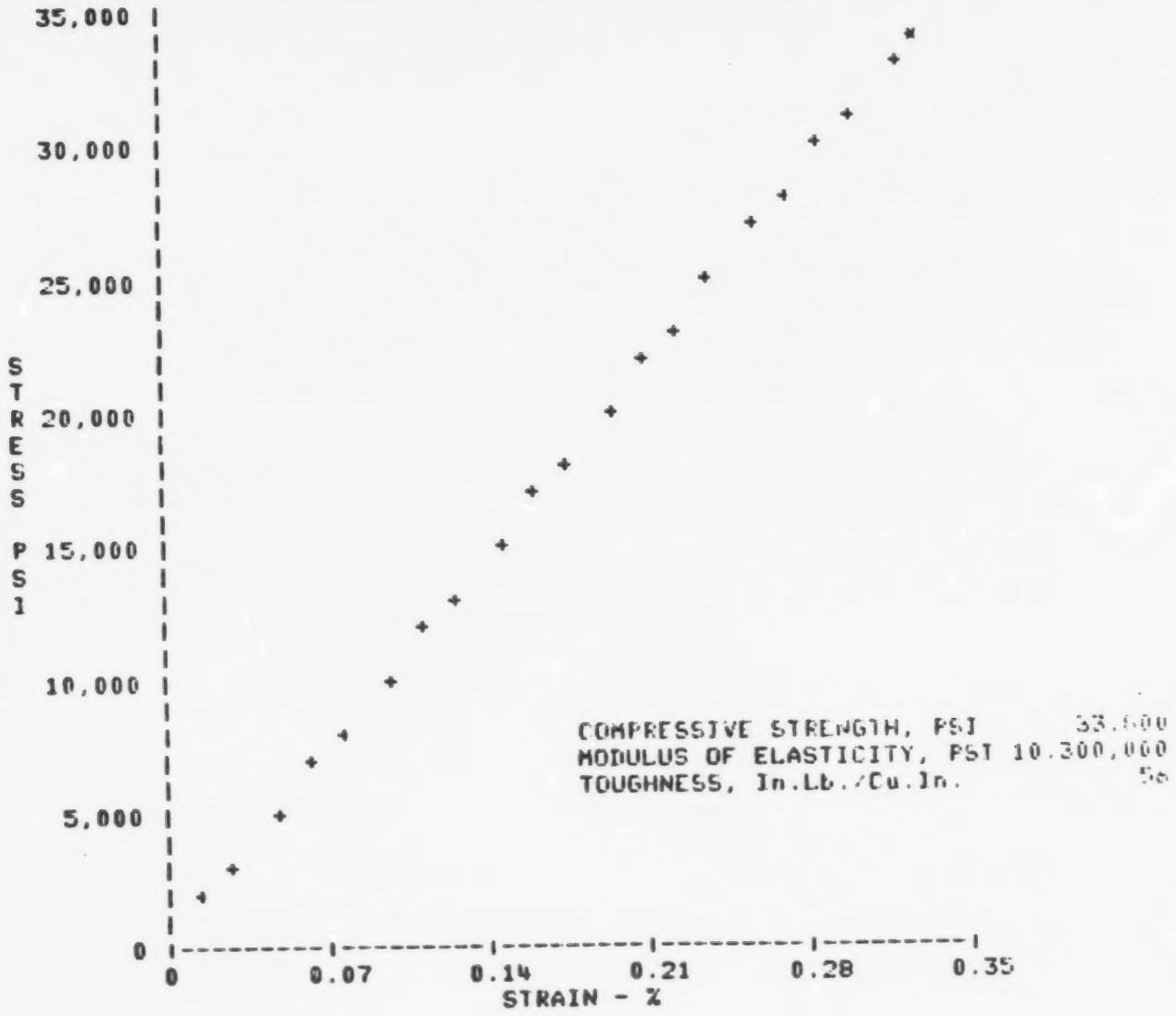


FIG. 1.10 UNIAXIAL COMPRESSIVE STRENGTH TESTING OF A ROCK SAMPLE

possible to optimize block caving since the surface required to initiate the caving of the blocks can be determined and their approximate dimensions can be evaluated. This classification can also be used in determing drilling patterns required for each rock class and locating zones of low competence (in order to avoid developing service drifts, storage areas, etc). Finally, the classification technique can also be used in the assessment of the cave angle of the rock mass.

The classification is not designed to solve all problems encountered but to provide extensive information regarding the rock mass being mined. The greater the knowledge regarding the rock mass, the greater the chances of controlling its behaviour.

This procedure has been employed with considerable success for several years and has shown that it is possible to modify and change the state of the rock mass by the application of a method based on ratings.

As is seen by the various measures developed by the management of the mine to control the stability of underground structures, it is possible to acquire much more information using diamond drilling than is normally done by the mining industry. In addition, the in situ evaluation of a rock mass is a dynamic process which involves all levels of mine personnel.

MINE 2

COPPER, GOLD AND SILVER MINES: CUT-AND-FILL MINING

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2.0 COPPER, GOLD AND SILVER MINES: CUT-AND-FILL MINING

2.1 BRIEF HISTORY OF THE MINE

The mining company owns several deposits, two of which have been in production since the early 1950s. Copper, gold and silver ore are mined in each of the deposits. Total annual production in 1973 was approximately 900 000 metric tons. The deposits occur in shear zones and alteration zones of a Precambrian anorthosite rock mass The principal types of variations encountered are carbonate rocks, sericite and chlorite.

The two deposits under production consist of mineralized veins extending from 60 to 250 m in depth, 30 to 240 m in length and 2 to 20 m in width. The dip of the veins varies between 35 and 80°.

The extraction levels are spaced at intervals of 46 m and, in one particular deposit, at intervals of 92 m, starting at a specific depth. The shafts have been sunk to -1 000 m and -1 188 m.

2.2 MINING METHOD

During the early years of production, the mining methods employed were cut-and-fill mining, shrinkage stoping and longhole inverted bench blasting.

Currently, only the cut-and-fill method is employed at the two deposits. It became necessary to fill the stopes due to increases in pressure in the deeper drifts. Cut-and-fill mining is more costly than the two other methods previously used but enables more efficient control of

dilution. The ore recovery rate is in the order of 95%. The size of the stopes generally corresponds to the dimensions of the mineralized veins.

A plan view and longitudinal section of the mining sequence of a given stope are illustrated in figure 2.1.

The excavation of the ore begins by the development of a sublevel 8 m above the lower drift. Ore is recovered from the pillar thus formed at the same time that the lower stoped is mined. Some stopes are mined from a drift located in the footwall at the same level as the stope itself. With this method, a pillar is eliminated and a greater volume of ore is recovered.

The hydraulic fill material consists of mill tailings which is passed through a cyclone in order to eliminate the finest tailings. This produces a silt-sand material. The average grain-size curve of the solid tailings transported underground is illustrated in figure 2.2. The tailings, which are mixed with water, are comprised of 60% solids and 40% liquids (% weight). Percolation occurs relatively quickly, at a rate of approximately 15×10^{-2} cm/s. It should be noted that certain stopes are dry-filled with granular material taken from borrow pits.

The first sublevel of each stope is filled with one part cement per eight parts filling material to a thickness of two metres. Hydraulic filling ensures greater safety conditions and decreases the dilution of the ore.

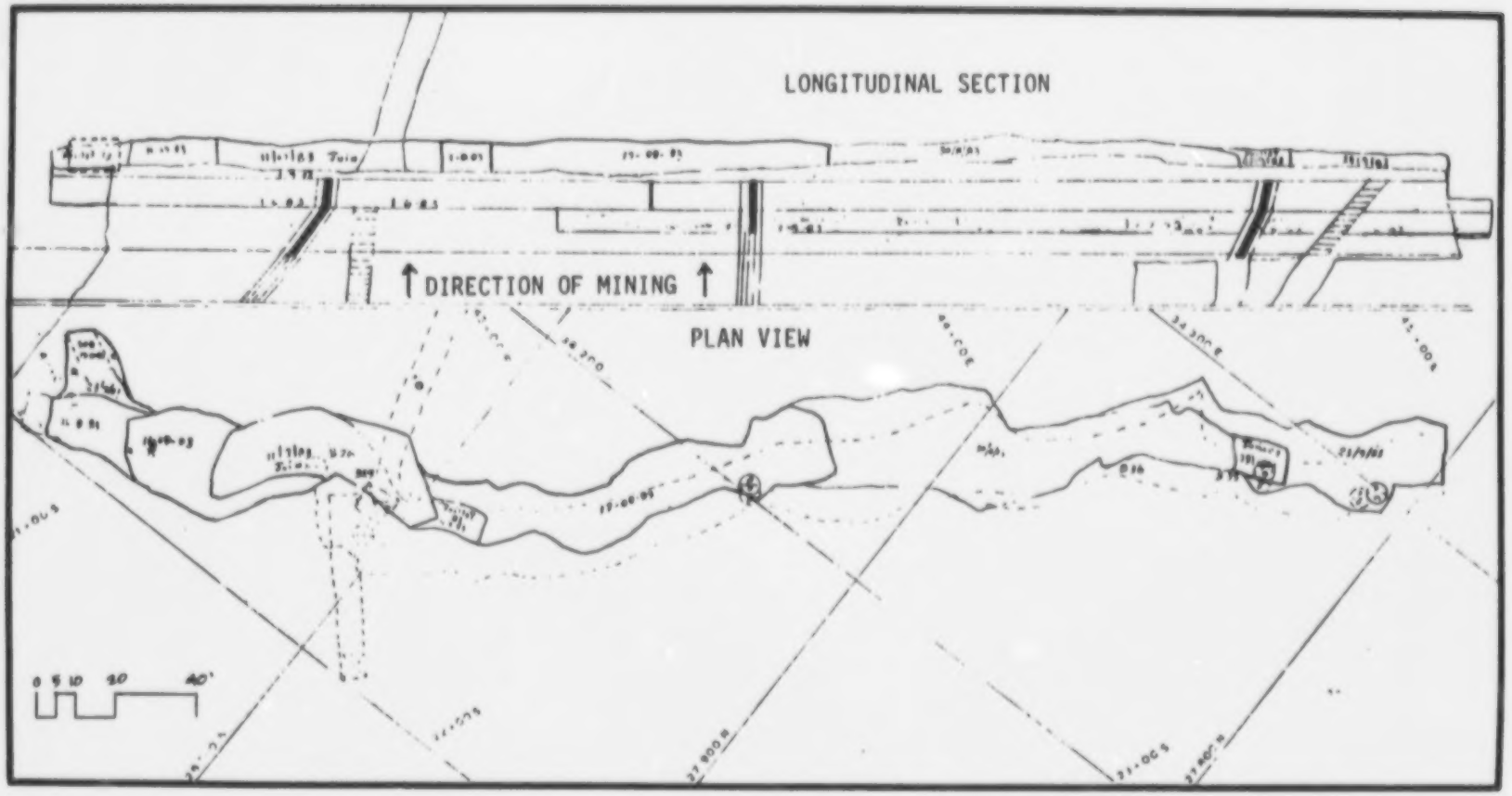

FIG. 2.1 MINING SEQUENCE IN A STOPE

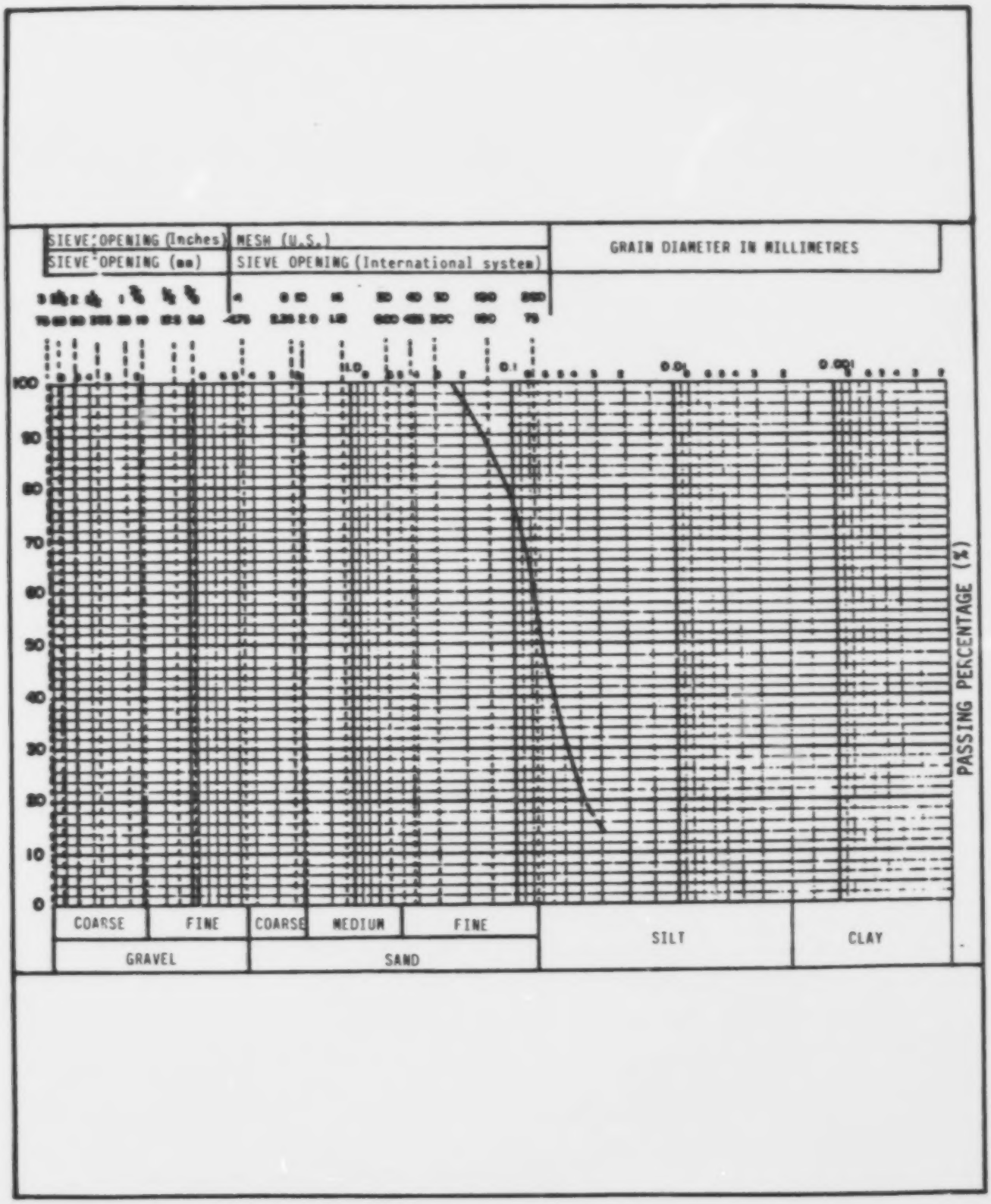

FIG. 2.2 GRAIN SIZE OF UNDERGROUND FILLING MATERIAL

2.3 TERMINOLOGY USED

Terms used in the cut-and-fill method include upper and lower pillars, temporary pillars and surface pillars. The upper pillar can be compared to the crown pillar which was described in Annex 1. The lower pillar refers to the sill pillar which is left in place in the bottom of the stope temporarily in order to protect the drifts, drawpoints, access tunnels, etc. The sill pillar is generally composed of ore which is recovered after the overlying stope is exhausted.

2.4 SURFACE PILLAR DESIGN CRITERIA

2.4.1 Study of surface and ground water conditions

Due to the presence of lakes above the deposits to be mined, the management of the mine has taken special measures in order to ensure the stability of the underground stopes. During the exploration phase, a number of drill holes traversing the hanging wall were filled with shotcrete in order to reinforce the rock mass and make it less permeable. This protective technique was reserved for drill holes which extended under the lakes and was essential due to the poor quality of rock encountered in certain parts of the surface pillar. The geological analysis of borehole cores reveals a heterogeneous surface pillar in which leaching zones occur. These zones are of lower ore grade and have poor mechanical properties; the action of water and chemical reactions which have taken place with the sulfurs have transformed certain areas of hard rock into granular, gossan-type deposits (figure 2.3).

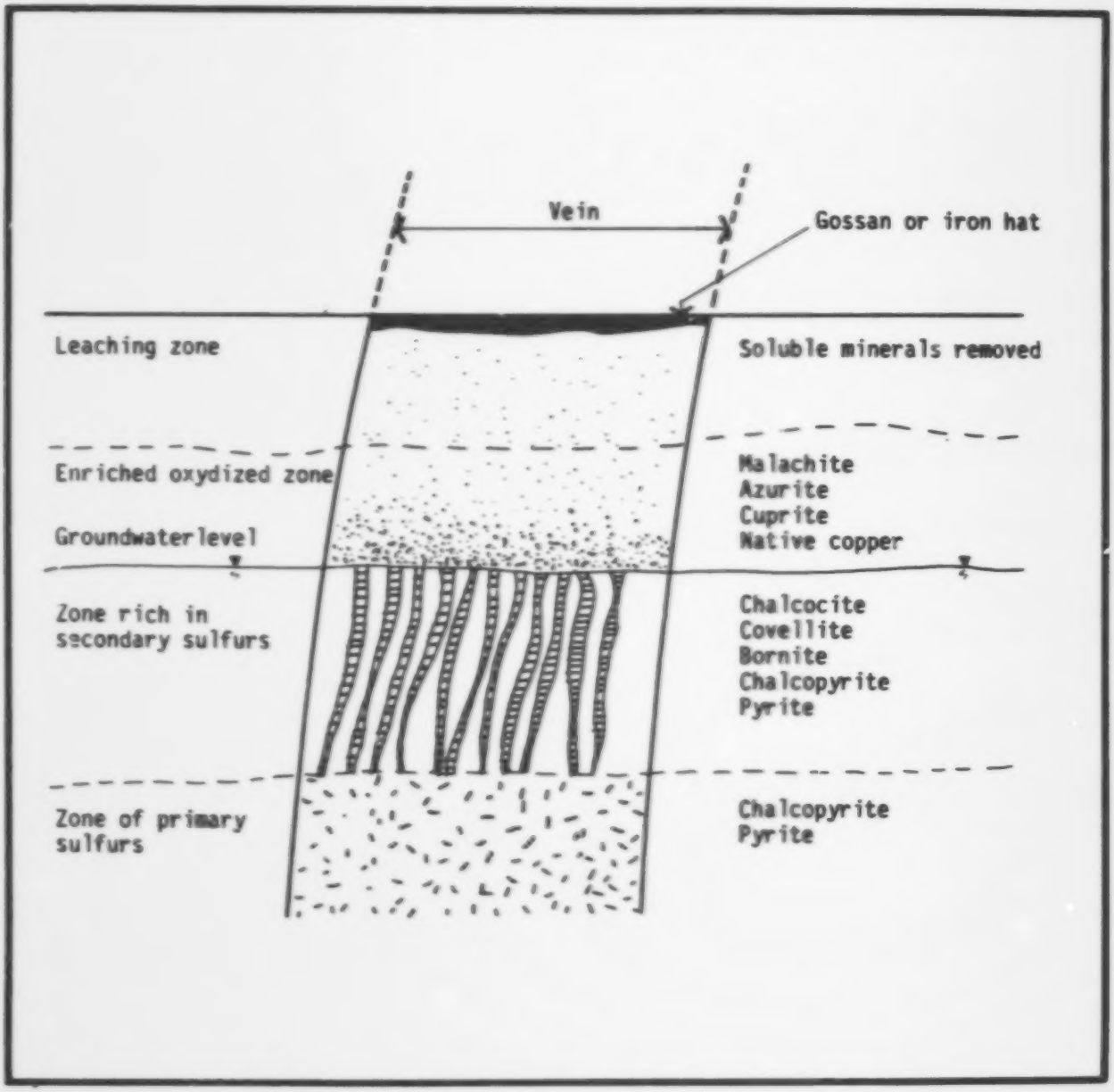

FIG. 2.3 Schematic representation of alteration zones in a sulfur vein (Taken from Mason and Berry, 1968)

The production of the mine and the continuous entry of oxygen into the upper stopes also contributed to the formation of gossan in the surface pillar.

Surface water, groundwater and water transported to the mine for hydraulic filling cause certain problems for the management of the mine. Mine production under dry conditions is above all a question of safety, profitability and productivity. In order to dry out the mine, the management has developed a very efficient pumping system to be employed at various levels of operational mines. From 0 to 40 m in depth, the pumps operate at a capacity of 80 to 90 L/s, while at greater depths, it may drop to 10 L/s. The substantial infiltration of water into the upper stopes is due to a large extent to faults and fractures in the rock mass and to leaching zones which have become more permeable.

2.4.2 Soils overlying the mineralized zone

Several surface pillars are covered with mine waste which is saturated with water. The stability of one of these pillars was considered critical by the management of the mine and consequently formed the subject of stability studies conducted on site by an engineering team and outside consultants. This study revealed that a 10-metre layer of overburden covered the pillar. Drill holes were made and various tests were conducted to establish the stratigraphic profile and geotechnical properties of the layers of sediment. Permeability tests were carried out in order to determine drainage conditions and piezometers were installed in drill holes to enable the monitoring of changes in the groundwater level resulting from surface and underground pumping.

2.4.3 Model of the mine

No computer model of the mine has been realized.

2.4.4 Rock mechanics

The management wishes to establish a rock mechanics program centred principally on the *in situ* behaviour of the rock mass. With this end in view, it recently hired a rock mechanics engineer whose main responsibilities consist in the collection and interpretation of field data.

The participation of mine personnel in such a program is essential if the program is to be effective. The foremen prepare reports on the stability of the rock mass and the conditions of the stopes at each work station. Figure 2.4 indicates the type of information gathered by the foremen in their reports.

With the objective of controlling local stability conditions more efficiently, the management has authorized the realization of a study on the development of various methods and procedures which could be employed to monitor stability. The first phase is the constant recording of field data (the foremen's report). The management also intends to integrate structural data obtained from the mapping of drifts into its mining plans. This additional information will be useful in predicting the behaviour of the rock mass in the new stopes and drifts. The second phase of the program consists in applying a rock mass classification to assist in the delimitation of zones which have a wide range of competence and which characterize the mine sites.

RAPPORT JOURNALIER DE POSTE SOUS TERRE

MINE: ____ _____

POSTE _____

DATE: ____ / ____ / ____
AN MOIS JOUR

EMPLOYE		NUMERO DE COMPTE	HEURES								TAUX HORAIRE TEMPORAIRE	CONTRAT	LIEU DE TRAVAIL	EQUIPEMENT		MINAGE				
NUMERO	NOM		1- REG 2- SUPP. 3- DBL 4- STA 5- URG 6- DIM.											GENRE	NUMERO	TOT PIEDS FORES	1- ELARGISSEMENT 2- FRONT DE TAILLE 3- VOLEE	NBR. DE TIRS	JOZEGOMZE	NBR. SAUTE
				01 FORER ET FAIRE SAUTER	02 RACLAGE	03 BOISAGE	04 CHARROYAGE	05 REMBLAYAGE	06 SERVICE											

(Rock mass stability)

EXPLOSIF			TRANSPORT			BOULONS				CABLES			RESERVE POUR LE COMMIS						REMARQUES
ANFO 100%	ANFO 60%	GELATINE	CAPACITE VOLUME	NOMBRE MINERAL	NOMBRE STERILE	1- A EXTENSION 2- SPLIT SET 3- SWELLEX CODE	3'	5'	7'	LONGUEUR	QUANTITE		1 CODE	1 QUANTITE	2 CODE	2 QUANTITE	3 CODE	3 QUANTITE	

FIG. 2.4 INTEGRATION OF INFORMATION ON ROCK MASS STABILITY IN FOREMAN'S DALY REPORT

2.4.5 Support

The roof and walls are supported by traditional bolting systems. Four types of bolts are used: expansion shell bolts, split-set stabilizers (friction), Swellex (friction) and resin-anchored bolts. Cables and resin-anchored bolts are used in special cases in which the rock is of low competence or the stope is relatively wide.

The filling of the stopes constitutes an excellent method of stope reinforcement and the cementation of certain pillars increases the stability of underground workings.

Experience has shown that openings of less than 6 m in width require little or no anchor bolting whereas larger openings often require the use of a systematic bolting system. In very large stopes, the bolt pattern may be 3 m x 3 m with holes of 18 m in depth.

The comments and suggestions of miners regarding the support program are carefully studied by the management.

2.4.6 Instrumentation and control

All stopes are inspected monthly including those which are mined out and situated beneath the lakes. The upper stopes are filled to within two metres of the roof, leaving access for inspection. The various stopes are inspected in the order indicated and enumerated in figure 2.5. All change in conditions are noted and fallen rocks are indicated with paint in order to monitor the deterioration of the pillars. An investigation is conducted on any significant collapse which occurs. Monthly load cell readings are taken and the groundwater level is monitored by means of piezometers.

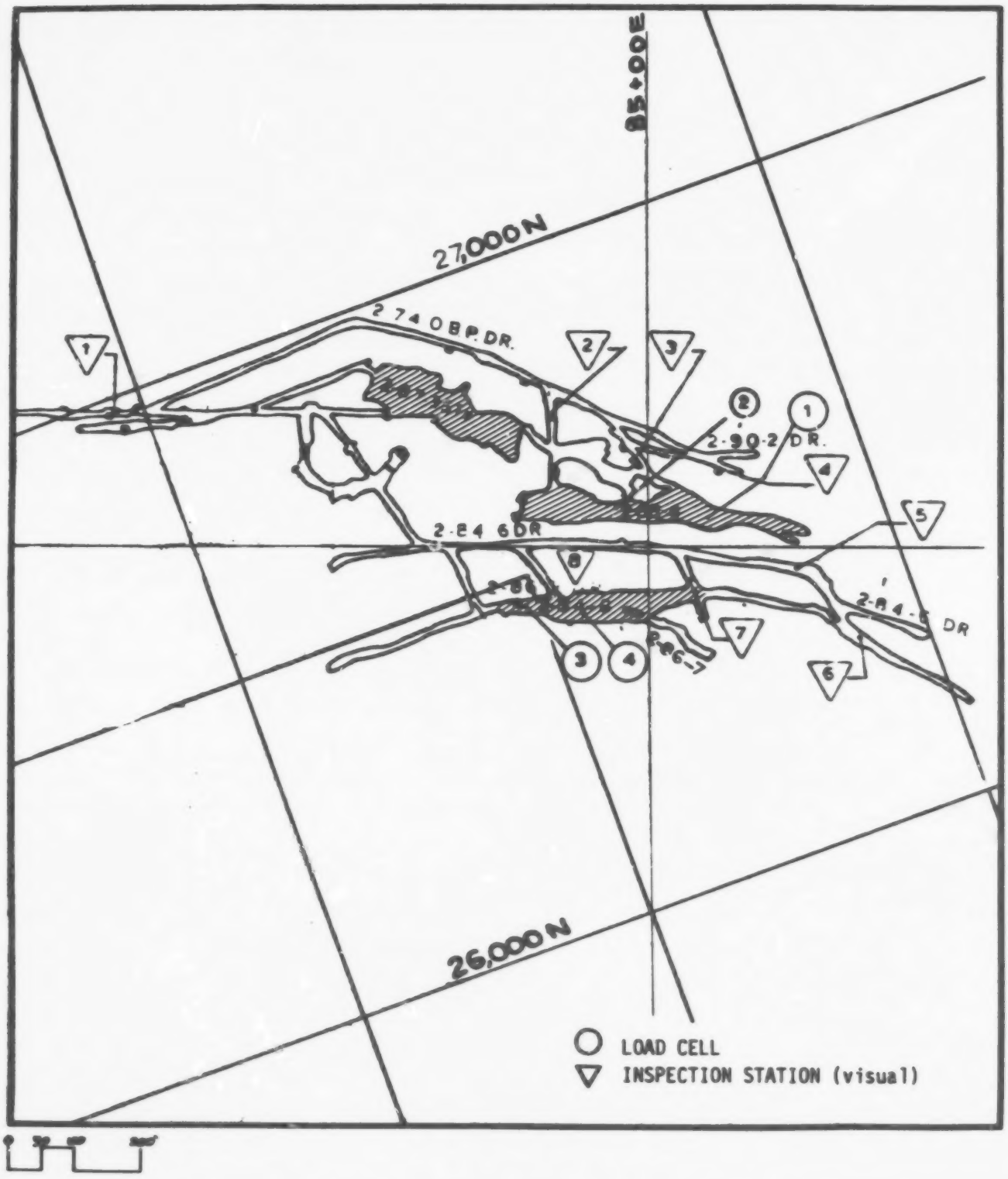

FIG. 2.5 PLAN ILLUSTRATING THE INSPECTION STATIONS AND SURFACE PILLARS EQUIPPED WITH LOAD CELL

The stability of the surface pillars is increased by means of the cement screen in the footwall, the bolting of the upper stopes, continuous pumping of infiltrating water and the filling of stopes.

2.4.7 Geometry of surface pillars

The dimensions of the surface pillars and underlying stopes are indicated in table 2.1. A horizon of overburden, varying from 3 to 7 m in thickness, covers these surface pillars and a layer of water, or mine waste saturated with water from a lake situated in proximity to the tailings disposal site, overlies the overburden.

2.4.8 Rock mass evaluation

The objectives of a study commissioned by the management in 1981 were to review and compare various methods employed by small underground mines to evaluate local stability conditions and to develop an internal technique for systematically evaluating slope stability. Although this study has not yet been completed, we were able to obtain a copy of a report from the management of the mine describing the first steps undertaken. The methodology followed can be summarized as follows:

i- development of a system for recording stope stability data (foreman's report mentioned in 2.4.4);

ii- evaluation of the value of integrating structural and geotechnical data into the design of plans and sections of the mine (description of borehole cores, rock quality designation, load point index, stereographic analyses, etc.);

TABLE 2.1: DIMENSIONS OF SURFACE PILLARS AND UNDERLYING STOPES

	EXAMPLES						
SURFACE PILLARS	≠1	≠2	≠3	≠4	≠5	≠6	≠7
OVERBURDEN:							
- Thickness (m)	3	5	6	6	5	7	3
SURFACE PILLARS:							
- Thickness (m)	59	49	49	49	58	17	56
STOPES:							
- Length (m)	61	46	61	107	76	84	79
- Width (m)	5	21	9	9	12	5	6
- Height (m)	46	46	46	43	40	55	50
REMARKS:							
SURFACE WATER							
- Thickness (m)	1	12	11	11	12	-	2
SURFACE LAYER OF TAILINGS							
- Thickness (m)	-	-	-	-	-	2	-

iii- determination of the applicability of current rock mass classification techniques to the local conditions encountered in the mine (description, advantages and disadvantages of the classification schemes of the Norwegian Geotechnical Institute (NGI) and the Council for Scientific and Industrial Research (CSIR)).

Phase " i" corresponds to the information gathered by the foremen of the underground stopes, as outlined in figure 2.4. This procedure enables the detection and evaluation of changes in stope stability conditions and increases the degree of awareness of the management and mine personnel regarding these conditions.

The second phase examined regards the integration of additional data into the mining plans, such as the rock quality designation or load point index, which should permit a more detailed evaluation of the rock mass and the development of a more realistic approach to the design of mining plans. The authors of the report point out that the values or indices provided by the load point test can be used to evaluate the uniaxial compressive strength of the rock mass using the following formulas:

$$Sc = (14 + 0.175\ D)\ I_S \qquad \text{(Bieniawski, 1975)}$$

where S_c = uniaxial compressive strength (estimated)
D = diameter of the sample (borehole core)
I_S = load point index

$$I_S = \frac{P \text{ (load applied to fracture sample)}}{D^2 \text{ (diameter of the sample)}}$$

The size of cores used in the load point test must respect the following condition:

$$\frac{\text{length of core}}{\text{diameter of core}} \geq 1.5$$

In order to more accurately compare the quality of rock at various locations in a rock mass, mining companies are adopting, to a greater and greater extent, classification techniques based on ratings assigned to parameters which affect the behaviour of the rock mass. This permits the determination of potentially unstable zones and facilitates the design of mining plans.

In the third phase of the study, two classifications - that of the Council for Scientific and Industrial Research (CSIR), developed in South Africa, and that of Norwegian Geotechnical Institute - are used by the authors who consider them to be highly sophisticated and able to characterize the types of rock found in Canada. Both of these classifications are based on a set of paramaters as outlined in table 2.2.

With the CSIR classification, rock mass quality is the sum of the ratings assigned to each of five parameters whereas in the NGI classification, rock mass quality is calculated as follows:

$$Q \text{ (rock mass quality)} = \frac{RQD \times Jr \times Jw}{Jn \quad Ja \quad SRF}$$

TABLE 2.2: PARAMETERS ASSESSED IN THE CSIR AND NGI CLASSIFICATION SCHEMES

CSIR CLASSIFICATION	NGI CLASSIFICATION
1) Uniaxial compressive strength (can be estimated using index)	1) R.Q.D.
2) R.Q.D.	2) Joint sets = Jn
3) Joint spacing	3) Joint roughness = Jr
4) Condition of joints	4) Alteration of joints = Ja
5) Water conditions	5) Joint water = Jw
	6) Stress reduction factor = SRF

The disadvantage of the CSIR classification is that a uniaxial compressive strength test must be conducted in order to determine the quality of the rock. If this test is not carried out, the uniaxial strength can be approximated using Bieniawski's formula (1975) and the stress index mentioned above. The values of the four other parameters are determined through visual observation of the rock.

No test is required under the NGI classification system but adequate knowledge of the description of the joints must be acquired, after which the application of the method becomes simple according to the authors of the report.

If the rock quality designation is unknown, it can be evaluated using the following empirical formula developed by Hoek and Brown (1980):

$$R.Q.D. = 115 - 3.3\ jV$$

Where jV = number of joints per m^3

The authors of the report have studied and classified more than 150 samples or sites using the CSIR and NGI classification schemes. The correlation between the values obtained by each classification is relatively strong ($r = 0.74$). The equivalence relation calculated by linear regression is:

$$CSIR = 50.81 + 12.11 \log NGI$$

The authors of the report conclude that the inclusion of structural and geotechnical data (rock description) in plans and sections would contribute substantially to the understanding of the behaviour of the rock mass and that the additional cost of adopting this procedure would be relatively low. They feel that if would be easier to integrate the NGI classification into the rock mass investigations, particularly since no tests are required. Having conducted 168 uniaxial compressive strength tests, the authors feel that a relation exists between compressive strength and the classification developed by the company's geologists. Under this classification, the rocks of the mine are divided into nine groups, each of which represents a higher degree of alteration.

2.4.9 General

Several years ago, the management initiated a study on surface pillars, consisting of structural mapping, a land survey, and the preparation of detailed sections and plans of the pillars in question. These works enable a follow-up of the situation and inspection rounds ensure worker safety. Figures 2.6 and 2.7 illustrate several surface pillars and underlying stopes.

One interesting fact regarding the recovery of a surface pillar should be mentioned. During the extraction of ore from a surface pillar of approximately 40 m in thickness, deterioration of the roof occurred several hours, forming a chute of approximately 18 m in height and 3 m in diameter (figure 2.8). The stope was filled and immediately closed, the chute was plugged with cement and wood structures and the roof of the stope was reinforced with resin-anchored bolts. The deterioration was caused by the poor quality of the rock mass which produced leaching zones in this location and by the presence of two

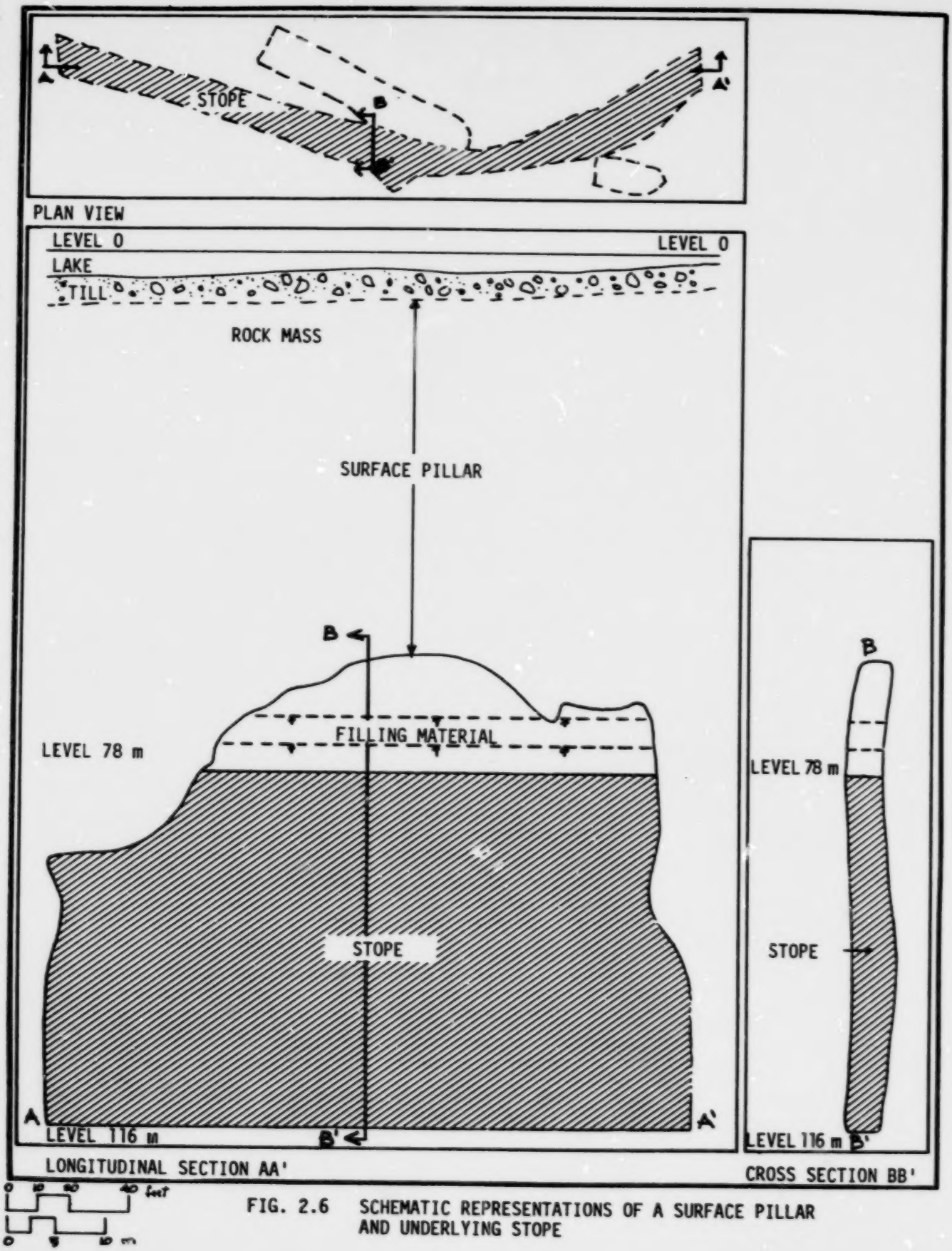

FIG. 2.6 SCHEMATIC REPRESENTATIONS OF A SURFACE PILLAR AND UNDERLYING STOPE

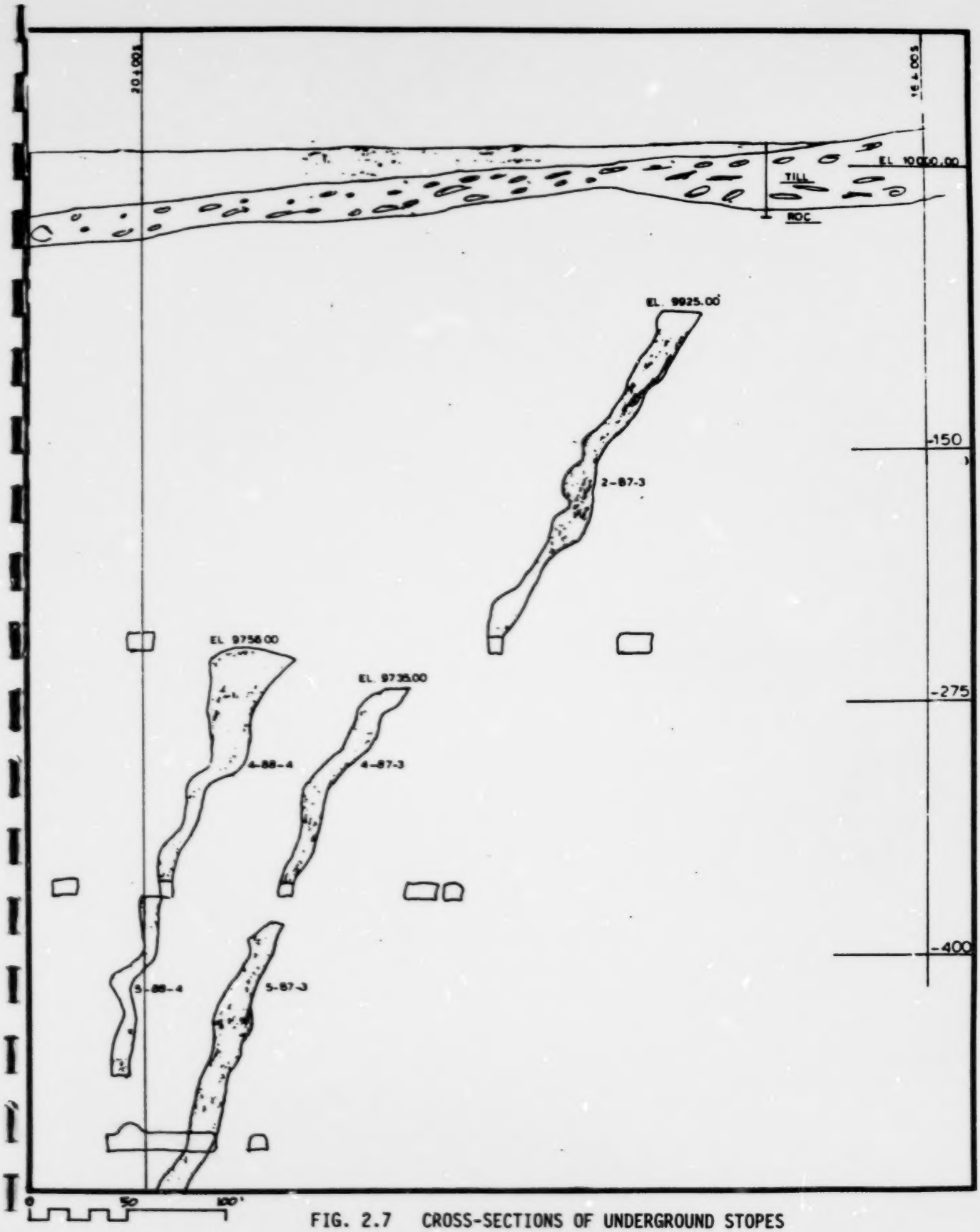

FIG. 2.7 CROSS-SECTIONS OF UNDERGROUND STOPES

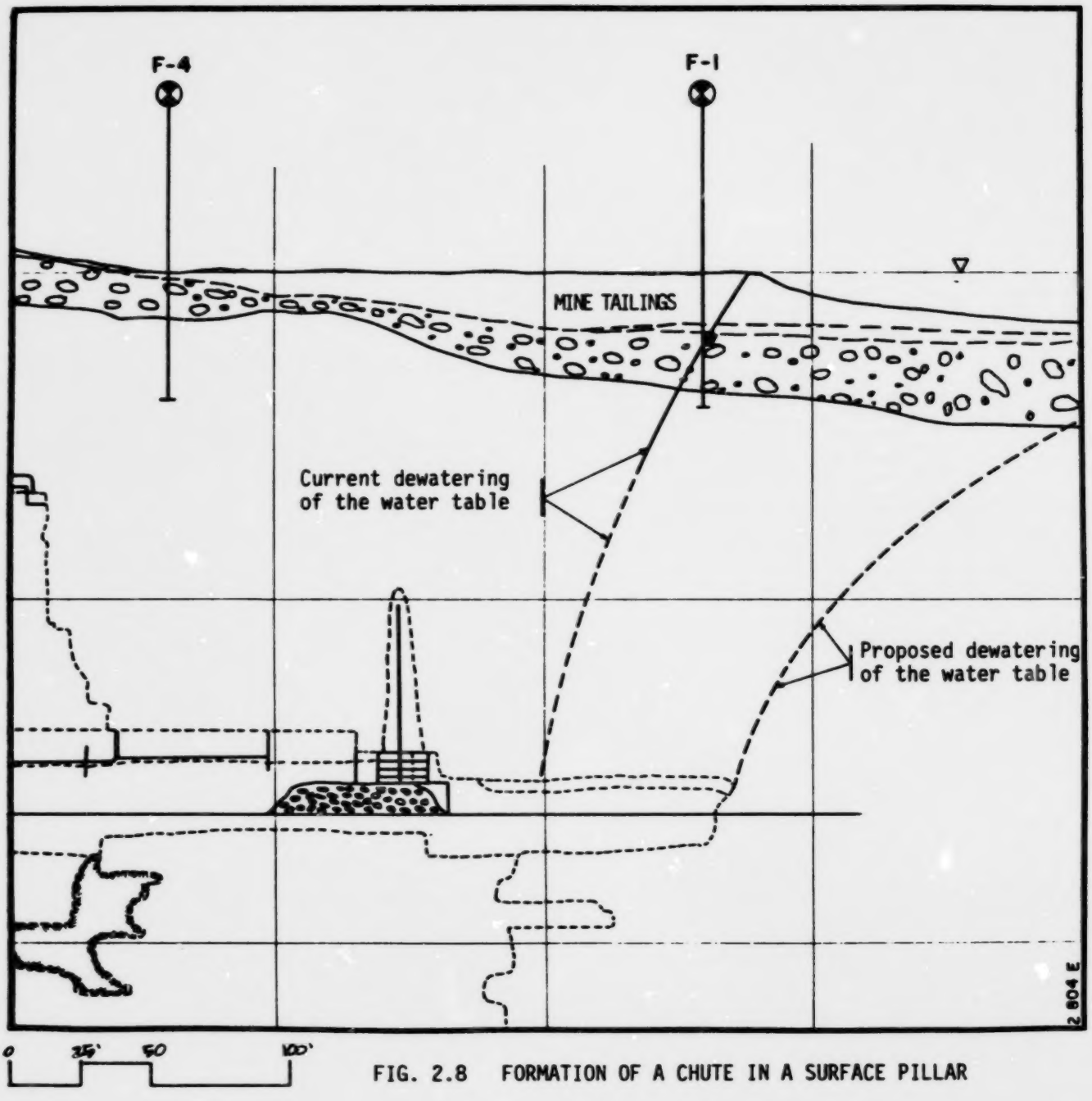

FIG. 2.8 FORMATION OF A CHUTE IN A SURFACE PILLAR

local faults which determined the extent of the collapse.

2.5 ASSESSMENT OF THE VISIT

During the initial phases of production, the approach adopted by the management was simple: the thickness of the surface pillars was arbitrarily determined in an intuitive and conservative manner. When the stopes reached the specified height the surface pillars were mined with much more care and their behaviour was monitored much more closely.

An important measure (injection of concrete into the drill holes) was taken by the management at a very early stage in the development of the mine in order to increase the quality of the rock mass of the footwall and decrease the infiltration of surface water, This decision was directly related to the presence of surface water. Through experience, the management already foresaw at that time potential problems which could arise during the mining of these deposits. Furthermore, although a formal rock mechanics program was not implemented, data regarding the structural aspect of the rock mass wera carefully gathered by geologists. The interpretation of these data enabled management to adopt a more realistic approach to the design of future mines. Geological data encompasses not only the grade, mineralogy and mapping of mineralized zones but also includes discontinuities and other factors which affect the stability of the rock mass.

Studies have shown that the rock mass overlying the upper stopes is of a heterogeneous nature in areas in which zones of hard rock intersect zones of leached rock having a soil-like consistency. Secondary mines and rock of low competence characterize these zones.

The periodic visit of stopes underlying the surface pillars and load cell readings have become well-established practices. Furthermore, the management has implemented a rock mechanics and stope stability program. Directors of the program are attempting to increase the awareness of all involved by taking a number of concrete measures such as including foremen in the evaluation of the stability of their own stopes or the mapping of discontinuities in drifts.

MINE 3

COPPER, GOLD AND SILVER MINES: BLASTHOLE STOPING, SHRINKAGE STOPING AND CUT-AND-FILL STOPING

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3.0 COPPER, GOLD AND SILVER MINES: BLASTHOLE STOPING, CUT-AND-FILL STOPING

3.1 BRIEF HISTORY OF THE MINE

The deposit was discovered in the 1920s, and the first shafts were sunk in the 1930s. For economic reasons, the mine was closed after only two years of production and was not reopened until the early 1950s. The deposit has been mined in three different zones. For the last twelve years, annual ore production totals approximately 1 000 000 metric tons.

The ore deposit consists of a series of chalcopyrite-bearing quartz veins that occupy fracture systems in extensively faulted and folded ultramafic/mafic Archean sills comprised of gabbro and pyroxenite. Approximately 75% of the mineralization occurs in the gabbro with about 25% in the pyroxenite.

The mineralized veins vary from 30 to 300 m in length, 1 to 6 m in width and 30 to 730 m in thickness. The veins dip between 55° and 70°.

Extraction levels are spaced at approximately 50-metre intervals and sublevels are spaced at 18-metre intervals. The lowermost level of the mine is -1 012 m in depth.

3.2 MINING METHODS

The greater part of the ore is currently extracted using blasthole stoping. Snrinkage stoping and cut-and-fill mining are also used. The ore is raised through three shafts.

The dimensions of the stopes correspond to those of the mineralized veins. The development of stopes in which blasthole stoping is undertaken began with the preparation of a 1.8 m X 2.4 m raise at one end for the development of sublevels. Where relatively narrow veins ($\leq$ 8 m) occur, sublevels were mined across the entire width for the drilling of the bench; for wider veins, bypass drifts were developed in the hanging wall and footwall in order to permit the recovery of large quantities of ore. Figure 3.1 illustrates the blasthole stoping technique. As shown in the figure, steel rebars were installed in a corner of the drift where stresses are concentrated. In order to more effectively control stope stability, pillars were left in place to separate the hanging walls and footwalls.

Shrinkage stoping is the second most frequently used mining method. With this method, ore is excavated in horizontal slices, starting from the bottom of the stope and advancing upwards. Part of the broken ore is left in the mined-out stope where it serves as a platform for mining the ore above and to support the stope walls.

The cut-and-fill method consists in drilling drifts in the veins and mining across its entire width, if the vein is narrow; in the case of larger veins, a haulage drift is developed in the footwall.

This procedure is more economical and more practical. Moreover, the scaling of rocks occurs less frequently with this method. The backfilling of stopes ensures greater stability of the rock mass and prevents the scaling of the walls which have been covered with backfill material.

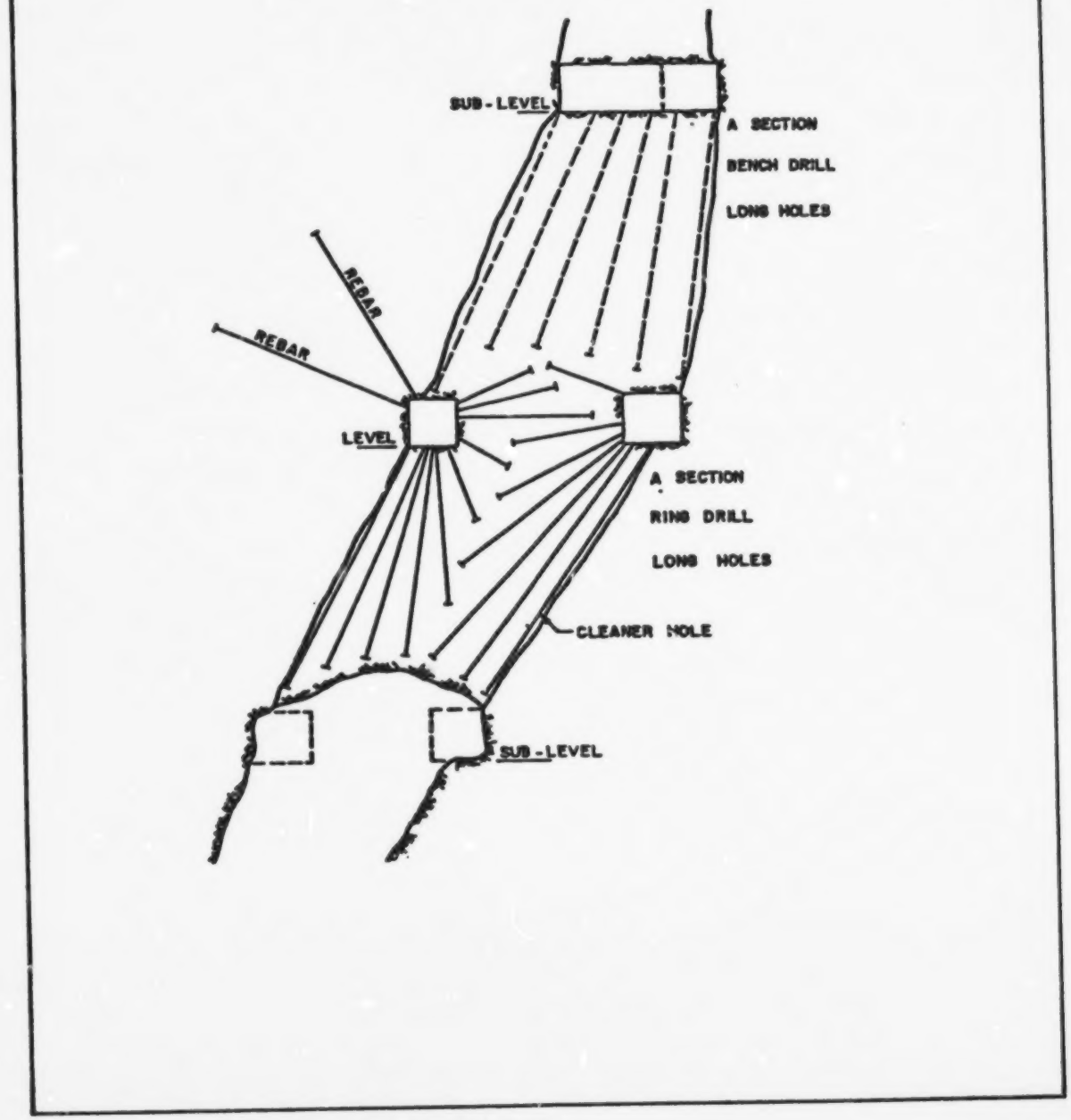

FIG. 3.1 LONGHOLE DRILLING AND BLASTING

Veins located under the stopes which have been mined using the cut-and-fill method are recovered in vertical trenches and supported by means of timber sets. The excavated areas are filled with a mixture of tailings and cement in a 30:1 ratio.

3.3 TERMINOLOGY USED

Pillars, permanent pillars, surface pillars and crown pillars are virtually the only terms related to pillars used by the management. In long, deep stopes, permanent pillars ensure the stability of the walls and floors by joining the hanging walls and footwalls. The terms surface pillars and crown pillars refer to the bedrock separating the overburden from the upper stopes of the mine. The term pillar is used most frequently and can designate both the crown pillar and the surface pillar.

3.4 SURFACE PILLAR DESIGN CRITERIA

3.4.1 Study of groundwater and surface water conditions

The entry of groundwater into the stopes is easily controlled by pumping systems installed in strategic locations. The level of water in the overburden is monitored by the use of piezometers installed in boreholes.

An in-depth technical study was recently conducted on a new deposit which is currently under production. Unfortunately, we were unable to obtain a copy of the report. However, a summary of the report indicates that a hydrologic study has been carried out. This study included the drilling of boreholes in overburden and rock mass, the conducting of permeability tests and the collection of various field data. The

conclusions of this study served as guidelines in the design of the pumping system.

3.4.2 Soil overlying the mineralized zone

Economic and technical studies are conducted when the overburden constitutes a potential danger to stopes situated close to the contact of overburden and bedrock. Boreholes are often used to determine the thickness of the overburden (figure 3.2). If it is not very thick, it is entirely removed or dried up in order to reduce the possibility of cave-in should it become so thick as to render the recovery of ore from it unprofitable.

In one particular case, problems related to the presence of water-saturated overburden overlying a surface pillar from which ore was to be partially recovered were examined in a study commissioned by the management. The various tests conducted on the overburden are outlined in a summary of the report. Granulometric and chemical analyses of samples revealed the presence of a sandy-silty soil of low cohesiveness, and the drilling of boreholes at 25-foot intervals enabled the topography of the bedrock to be determined. These boreholes indicate that no basins exist in which water could produce significant hydrostatic pressure. However, due to the low cohesiveness of the water-saturated soil, the management decided to take several preventive measures. These are described in greater detail in section 3.4.6 (Instrumentation and control).

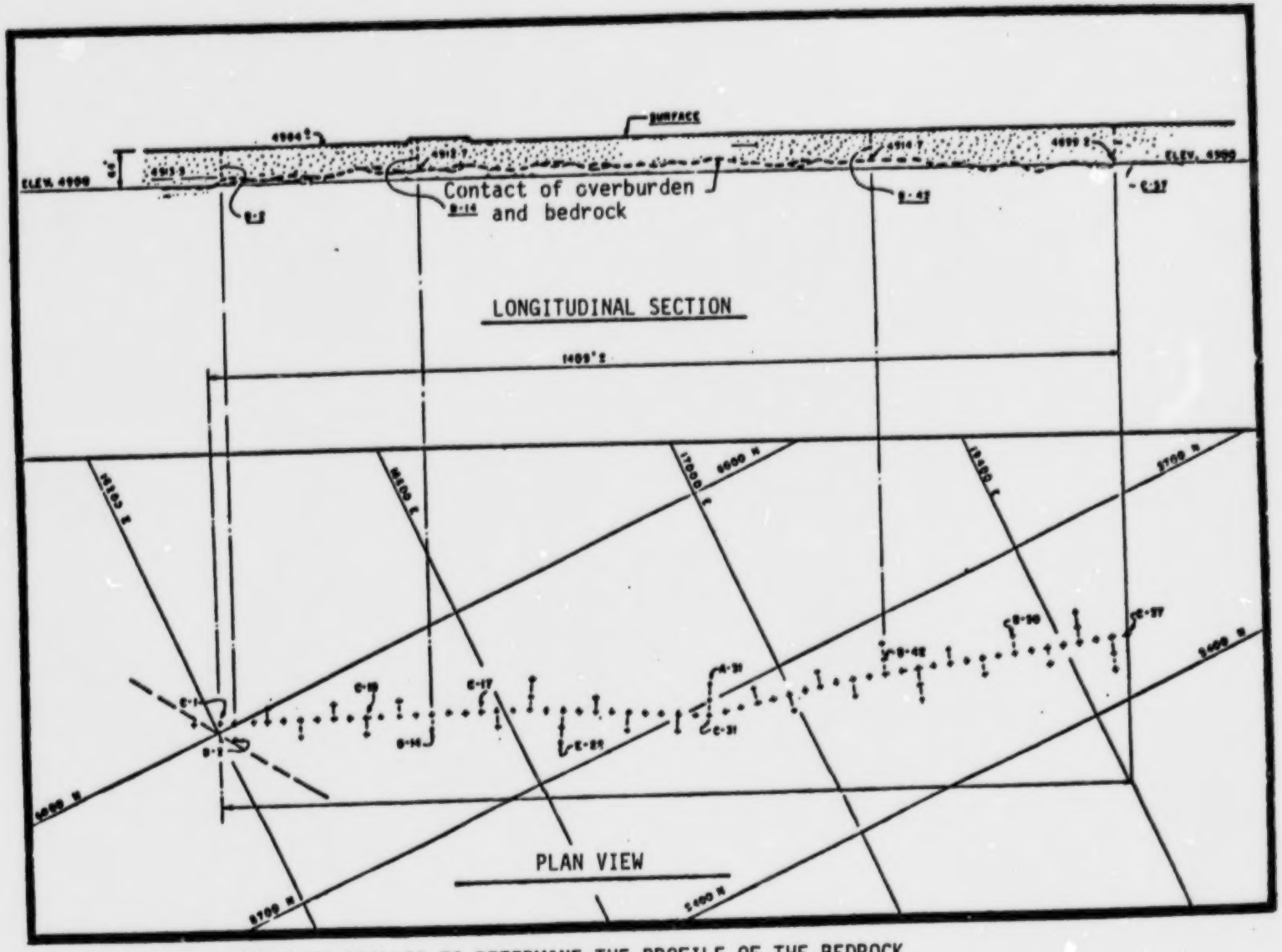

FIG. 3.2 BOREHOLES DRILLED TO DETERMINE THE PROFILE OF THE BEDROCK

3.4.3 Model of the mine

Computer models were applied within the framework of the study. Stresses and displacements were determine- by means of the finite-element method. A summary of the basic hypotheses and conclusions obtained from the analysis of one of these models was provided by the management of the mine. The model applies to a surface pillar from which the management wanted to recover part of the ore.

The mineralized vein had already been mined to a level of 55 m beneath the surface. The contact of rock mass and overburden was located at approximately 15 m, which left 40 m of rock mass above the stope. In this area, the vein varied from 2 to 3 m in width and approximately 150 m in length. The objective of the study in question was to determine the thickness of the surface pillar required under such conditions.

The study consisted essentially of the following three phases:

1- The gathering of basic data

2- The analysis and evaluation of stability

3- The determination of design criteria for a future pillar

The basic data required to conduct this study consisted of the following elements:

i) Analysis of the soil overlying the zone under study

ii) Strength and structure of the rock mass in question

iii) Interpretation of local geology

iv) Factors related to production

Data were gathered in situ, above and on either side of the future pillar, and in a laboratory. As mentioned in Section 3.4.2, the nature and depth of the overburden were studied in detail using boreholes drilled from the surface. Structural data were obtained from the mapping of lower levels which had already been mined and from the analysis of certain available borehole cores. The geo-mechanical properties of the various types of rock encountered were determined by the laboratory testing of representative rock samples.

The second phase of the project consisted of the evaluation of the data obtained in order to determine the elements to be taken into consideration in the design of the surface pillar. At this stage, the mathematical model was used. The finite-element method was employed to determine the stresses and displacements within and surrounding the future pillar. The data used for this simulation applied to four groups of rock: the mineralized zone, the surrounding rock mass, major shear zones and the overburden. Each group was defined by the following parameters which are commonly used in this type of analysis: uniaxial compressive strength, tensile strength, cohesiveness, angle of internal friction, density of the rock, modulus of elasticity and Poisson's ratio.

Although we cannot elaborate on the manner in which the analysis was conducted, we can mention that the uniaxial compressive strength used in the model to define the rock in which the mineralization occured and that of the neighbouring rock mass was approximately 20% of the average laboratory value obtained from borehole cores. The compressive strength value used for rock within shear zones was decreased to 5% of that of laboratory results.

We should also mention that the safety factor determined in this study was calculated by means of the ratio of the average uniaxial compressive strength of the rock mass to the maximum stress acting at the centre of the surface pillar, a value obtained by calculation.

The model was developed based on the assumptions that shrinkage stoping was used for widths of 2 to 3 m, and that underlying stopes at a elevation of -55 m were 4 m in width and completely mined out.

The considerable amount of information gathered and processed enabled the determination of significant design criteria. The major conclusion of the study is that, based on the hypotheses retained, it is possible to continue mining above the level of -55 m to an elevation of -27 m. This would leave a minimum surface pillar of 12 m, or 5 times the width of the unfilled stopes. The study concludes that the stability of this pillar would be long term and provides a general assessment of the following design factors:

i) The soil is completely saturated, of low cohesiveness and comprised of fine sand and silt; special attention should be given to vibration caused by blasting;

ii) The structure of the rock mass is such that it is divided into single blocks requiring the use of artificial support;

iii) The presence of water in the rock mass must be taken into consideration but does not represent a serious problem due to the loosening of the joints which should increase drainage;

iv) The study illustrates the sensitivity of the stability of the surface pillar as a function of the variation in its width. The study clearly indicates the location of potential fracture zones and provides an evaluation of the expected maximum displacement of the recommended pillar. The safety factor (according to the definition already provided) is set at more than 8 and the effect that filling the stopes would have on the thickness of the stope is assessed. In this case, the author recommends a 50% reduction in thickness if possible.

v) The author states that the monitoring of the pillar upon completion of mining operations is essential. He recommends the installation of instruments and the visual inspection of the openings.

3.4.4 Rock mechanics

Rock mechanics studies are generally conducted by outside consultants. The management does not have a rock mechanics division as such, but rather integrates such works into those generally carried out by engineers in the planning division. The planning engineers are responsible for determining the drilling patterns and the location of cables and equipment (extensometers, load cells, etc.).

Certain properties of rock samples taken from the orebody, hanging walls and footwalls are illustrated in table 3.1. The uniaxial compressive strength of these samples is relatively high with values of 139 and 154 MPa (20 200 and 22 200 psi).

TABLE 3.1 : ROCK PROPERTIES AND STRENGTH

LOCATION	UNIAXIAL COMPRESSIVE STRENGTH MPa	MODULUS OF ELASTICITY E, MPa	ROCK DENSITY KN/m³	NUMBER OF SAMPLES TESTED
Hanging wall	139	1.02×10^5	28.7	3
Footwall	154	0.96×10^5	29.2	3
Orebody	139	0.87×10^5	29.5	4

3.4.5 Support

All of the stopes are systematically bolted. Steel rebars of 2.5 cm in diameter are generally installed at intervals of 4 to 6 m. In cases in which the rock mass is of poor quality, cables and long resin-anchored bolts are installed. Where surface pillars are present, the following bolting pattern is used: a grouted steel rebar (20 mm in diameter and 2.1 m in length) is installed at every square metre of exposed surface beneath the pillar.

The stopes are generally filled in order to ensure greater stability of the wall and roof. They are divided into two sections by a pillar used to retain the fill material, which consists of a mixture of mine waste and hydraulic fill in a ratio of 6:1. As mentioned in section 3.2, certain stopes may be filled with a mixture of tailings and cement.

3.4.6 Instrumentation and control

The stability of the surface pillars is controlled by inspection rounds, and the stability of those parts of the stopes which are not situated beneath an accessible drift is monitored through the drilling of boreholes. These holes are filled with water to the collar; if the hole does not become filled, this is a sign that deterioration of the ground has occurred. Morever, if the entire length of one of the boreholes is not acessible, this could indicate movement of the rock mass.

During production, the mine is inspected weekly and upon completion of mining operations, rounds are made monthly. The frequency of inspections during the drawings of ore is once every other week and subsequently, once monthly. All unusual situations, such as a new flow of water or the formation of exposed fissures, are recorded and dated by the stope inspector, who is generally a planning engineer. He makes written comments on the situation and ensures the follow-up of the inspection.

Certain preventive measures which were taken should be mentioned. A mined-out stope was isolated from the rest of the mine by two 1.4 - metre thick reinforced concrete walls. These bulkheads were erected in case the surface pillar gives way, allowing water-saturated overburden covered by a marsh to enter. A pipe equipped with a valve runs through one of these walls, enabling potential entry of water into the isolated stope to be monitored. Moreover, several critical locations in which failure zones occur and certain stopes having critical geometric sections are reinforced with cables and very long steel rods.

Hydraulic filling of the underground stopes requires a large water supply. Fill time is a function of the permeability of the mine tailings. The rate of infiltration of the water through the fill material is estimated at 15 to 20 cm per hour. The installation of vertical drains in the tailings contributes to the drying of the filled stopes.

3.4.7 Geometry of the surface pillars

The average thickness of the many surface pillars in this deposit is approximately 15 m.

The dimensions of some of these pillars and that of the subjacent stopes are shown in table 3.2. Example 3 appears to present problems due to the presence of an overlying marsh. However, a detailed study of the overburden and surface conditions (minimum quantity of overburden, water, etc.) indicated that the underlying stope is not considered in danger. If should be noted that be stope in question is mined out as are all those listed in table 3.2.

3.4.8 Rock mass evaluation

No systematic rock mass evaluation program has yet been implemented by the management. However, in specific cases, the planning engineers evaluate the competence of the rock using boreholes and geostructural data gathered in the drifts.

In a special study commissioned by the management and conducted by a firm of engineering consultants, a rock mass containing a deposit under production was evaluated. This study was based on the description of borehole cores, the study of joints (more than 3 000) in various types of rock, laboratory testing (modulus of elasticity, Poisson's ratio, compressive strength, etc.) and mathematical models. The ratio of the horizontal stress to vertical stress (Ko) ranged betweem 2.5 and 3 and compressive strength values were calculated for -50, -150 and -300 m. The classification scheme adopted was that developed by Bieniawski (1974), the CSIR classification. The values obtained by this method ranged between 57 to 78, with an average of 70 to 75. These values designate a rock of high quality.

TABLE 3.2: DIMENSIONS OF SURFACE PILLARS AND UNERLYIND STOPES

SURFACE PILLARS	EXAMPLES								
	No. 1	No. 2	No. 3	No. 4	No. 5	No. 6	No. 7	No. 8	No. 9
OVERBURDEN:									
- Thickness (m)	14	3	4	4	--	--	--	--	--
SURFACE PILLARS:									
- Thickness (m)	8	12	9	14	6	8	9	21	23
STOPES:									
- Length (m)	38	61	30	43	--	--	--	--	--
- Width (m)	2	15	6	6	4	3	3	5	3
- Height (m)	46	91	107	--	--	--	--	--	--

COMMENTS:

Example 1 - Stope isolated by two concrete walls
- Marsh on surface

Example 2 - Overburden removed and pillar mined

Example 3 - Marsh on surface

Example 4 - Waste on surface

Example 5 - Bedrock on surface

Example 6 - Bedrock on surface

Example 7 - Bedrock on surface

Example 8 - Bedrock on surface

Example 9 - Gravel on surface

3.4.9 GENERAL

The management plans to completely mine several surface pillars as it has done in the past. The pillars chosen are situated in an area in which ground conditions can be easily controlled. The economic question is studied as a function of the nature and volume of the overburden, drilling results and quality of the rock mass.

3.5 ASSESSMENT OF THE VISIT

The approach adopted by the management in the face of problems related to mine production is both dynamic and safe. Preventive measures are modified depending on the behaviour of the rock mass, and new geostructural data are gathered during the various mining operations. Technical studies are periodically conducted by outside consultants retained by the management. These studies serve to a certain point as guidelines for decisions to be made regarding the development of the deposit. However, the final decision pertaining to the mining method to be used is made by the management. For example, it is the management which determines the upper extraction limit of a surface pillar, judging the risk factor involved. The degree of exposure of a miner to danger is one of the primary criteria in the selection of a mining method and determination of the order of operations.

The management has raised an interesting point regarding the application of the following safety standards imposed by an inspector of the Quebec Occupational Health and Safety Commission (C.S.S.T.):

For filled stopes or stopes and pillars: Thickness of the surface pillar ≥8 x width of stope

For unfilled stopes or open stopes: Thickness of the surface pillar ≥12 x width of stope

The management points out that in the case of intact rock masses, these standards may prove to be excessively stringent, resulting in the waste of considerable quantities of economically recoverable ore, and that in the case of extensively altered, heavily jointed, water saturated rock masses, these standards may not be severe enough. In other words, these standards do not take the nature and volume of the overburden overlying the pillar into consideration.

A suggestion made by the management of the mine is worth noting. They have proposed that a back analysis of pillars which have caved in with or without direct intervention of the operators would provide interesting details on fracture criteria and rock mass conditions. The principal limitation of this type of study, however, is the lack of data regarding previously recorded cases. If progress is to be made in this area, surface pillars should be equipped with instruments for purposes.

MINE 4

COPPER MINE (ZINC, SILVER): OPEN STOPE AND POST FILL METHOD

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4.0 COPPER MINE (ZINC, SILVER): OPEN STOPE MINING

4.1 BRIEF HISTORY OF THE MINE

This deposit was discovered relatively recently, in 1974. The decision to put the mine into production however, was not made until 1979. Annual ore production currently totals 500 000 metric tons. The deposit is located within a sequence of volcanic rhyolites, the principal mineralized zones being composed of chalcopyrites, pyrites and sphalerites.

One of the most important characteristics of the deposit is the presence of a major north-south fault, dipping 45° to 55° east. This fault forms a boundary along the lower contact of the deposit which is approximately 500 metres in length and between 15 and 45 meters in width. The vein is mined to a level of approximately -180 metres, following the dip of the fault. A general view of this deposit is illustrated in figure 4.1.

A layer of overburden, up to 45 m in thickness, overlies the deposit. Unconsolidated material, from the bottom to the top, consists of a layer of till, a sand and gravel deposit and a second layer of till, also covered with sand and gravel. The relative position of these layers of material is illustrated in figure 4.2.

One of the first major problems encountered during the development of this deposit was the presence in the overburden of a considerable quantity of water which could flow into future stopes. Initial estimates placed the water flow rate at up to 190 L/s. In order to deal with this problem the groundwater level was lowered at the start of

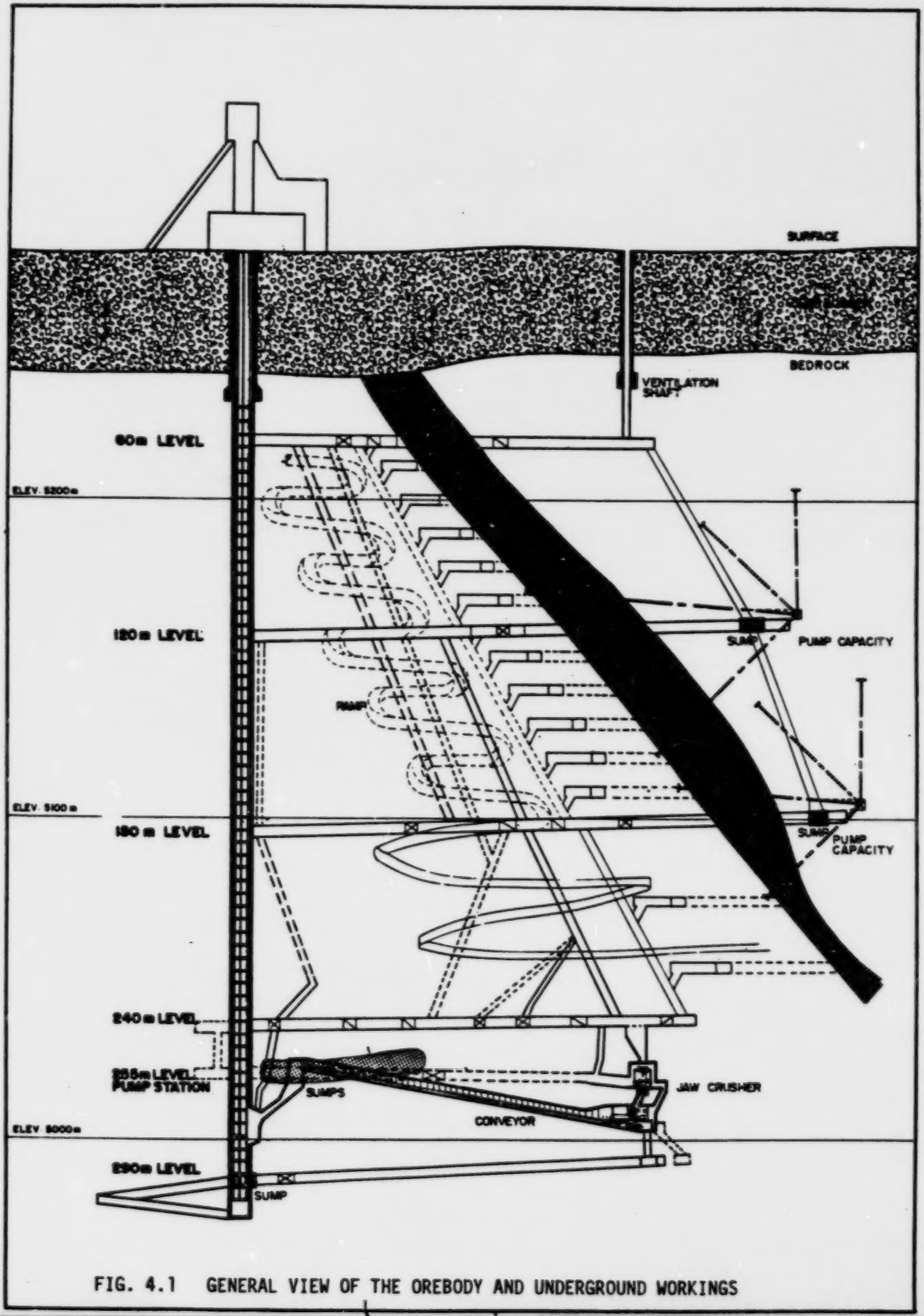

FIG. 4.1 GENERAL VIEW OF THE OREBODY AND UNDERGROUND WORKINGS

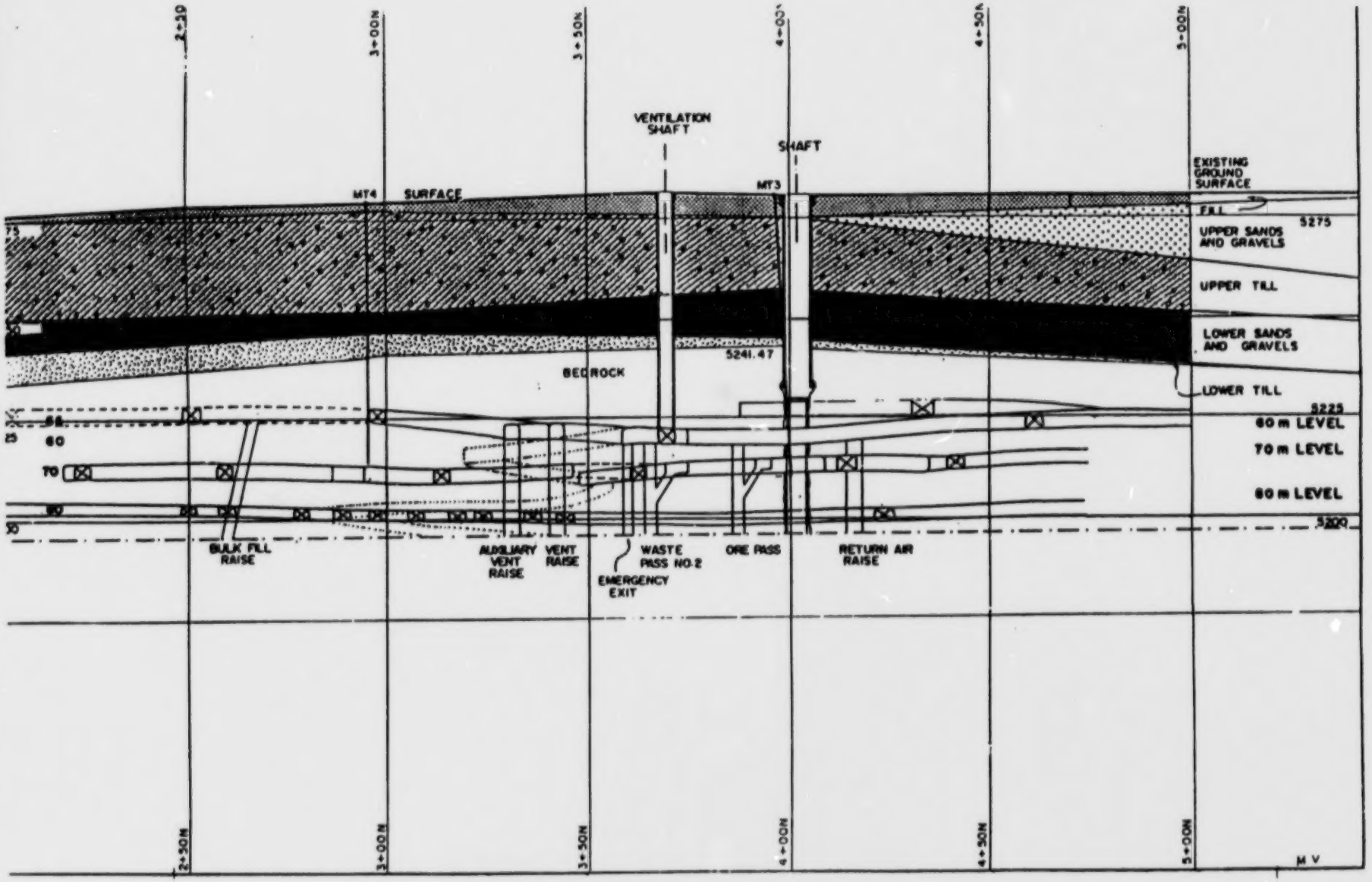

FIG. 4.2 DESCRIPTION OF THE OVERBURDEN

operations. This is discussed in greater detail in subsequent sections.

Finally, we should mention that extensive leaching has occurred in certain areas of the upper part of the mineralization, resulting in varying degrees of carbonatization and kaolinization of these locations. Very wide joints, measuring 20 to 30 cm, are present within the mineralized zone. These are filled with very fine sand transported by infiltrating waters.

4.2 MINING METHOD

Mining operations at this deposit are highly mechanized. Open stope mining (the stopes are filled after the ore has been entirely extracted through the bottom of the stope) is employed. Access to these stopes is provided by sublevels, developed from a service ramp every 20 metres. The main levels are spaced at 60-metre intervals.

Due to the shape of the orebody, mining operations began at a level of -180 metres. All drifts are situated in the footwall whereas the hanging wall serves for drainage and ventilation.

At each sublevel, the orebody is divided into 22-metre panels along strike and alternate panels are mined over the full ore width. Physical support is provided by filling the stopes. During initial operations, a mixture of cemented, classified mill tailings (1:20 cement ratio) was used. Today, fill material consists of hydraulic and waste backfill mixted with cement (1:20 cement ratio).

The ultimate objective of the management is to reach an ore recovery rate of 100%. It has already begun to extract ore from several pillars at lower levels. The surface pillar has not yet been formed; operations have reached the -100 m level and will continue to the -80 m level; this will leave a temporary surface pillar of approximately 35 m in thickness since the contact of overburden and bedrock is located at -45. This future pillar at level 55 m is illustrated in 4.3.

4.3 TERMINOLOGY USED

This company employs relatively few terms to refer to pillars. Because of the mining method used, the personnel uses terms which are commonly used in the North American mining industry. The stopes at each level are separated by pillars. The term pillar is used alone but designates barrier pillars. With the exception of the crown pillar, there are no horizontal pillars at this mine.

The operators of the mine use the term crown pillar or " stot" in French to refer to the uppermost section of the orebody. In this sense, the term encompasses the rock situated immediately above each of the upper twenty stopes of the mine. To the operators, the crown pillar is a continuous element of the same width as the mineralized vein.

4.4 SURFACE PILLAR DESIGN CRITERIA

4.4.1 Study of groundwater and surface water conditions

As early as the initial stages of exploration drilling, the serious

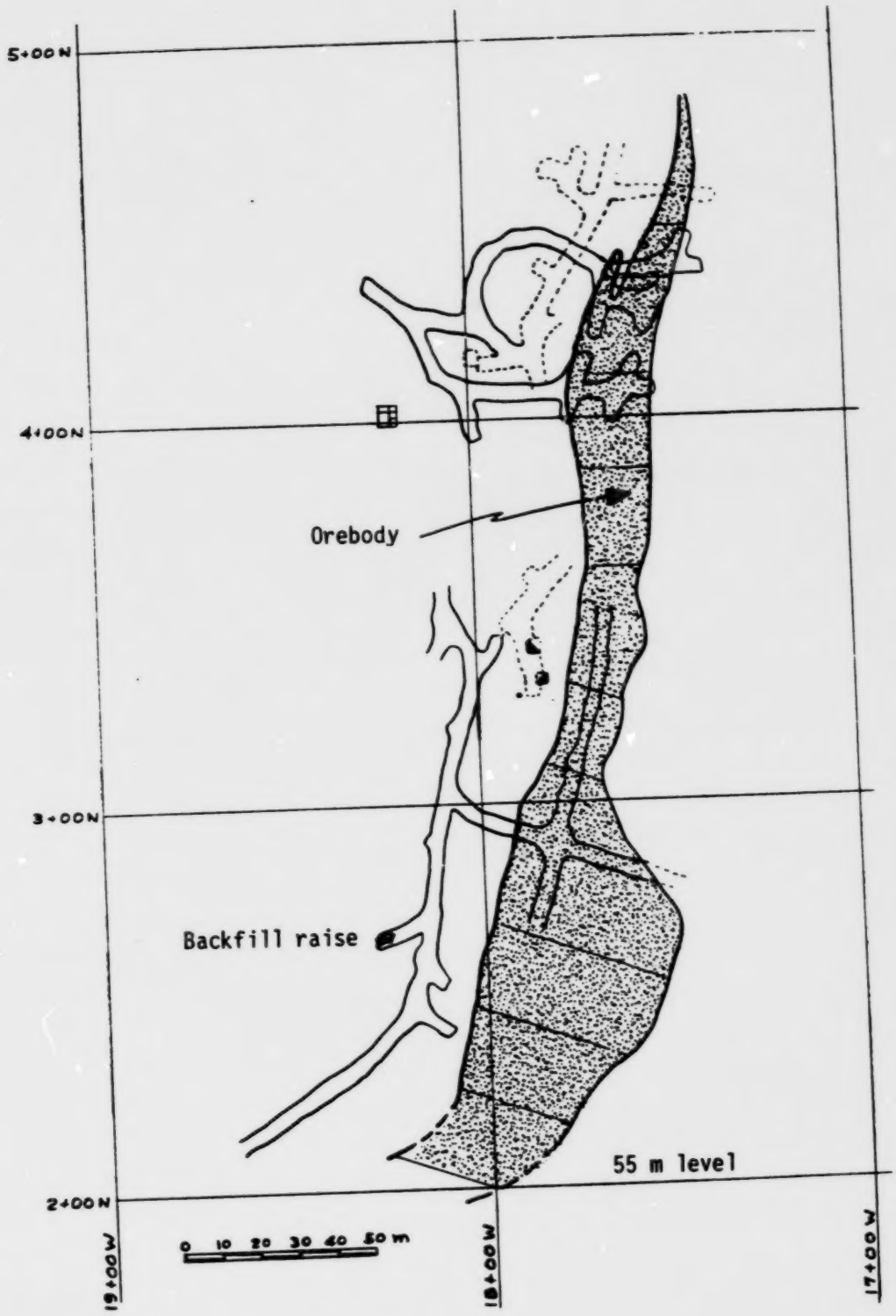

FIG. 4.3 SECTION OF THE SURFACE PILLAR

mining problems which could arise as a result of the particular hydrogeological conditions of the property became apparent. In fact, in this area, the groundwater level was located 5 m beneath the surface of the overburden which was more than 45 metres thick.

In the following pages, we shall summarize the evaluation and drainage techniques employed since the start of operations. Although the main objective of drainage is to improve ore extraction conditions, it also constitutes a major component in surface pillar stability.

The sequence of operations realized within the framework of hydrogeological studies are as follows:

- Determination of the groundwater level by means of exploration drilling (diamond drilling);
- Realization of pump and pressure packers tests in order to determine the permeability coefficients of the various aquiferous layers and rock types;
- Simulation and evaluation of the effect of underground operations on the flow network;
- Realization of falling head tests and installation of piezometers at various depths;
- Flow measurement from accessible underground development drifts. The results of these tests were used to re-calculate hydraulic conductivity values for various zones of the rock mass;
- Collection of data and re-evaluation of the hydrogeological cycle. Realization of further studies and further piezometer installations to monitor overburden and groundwater movement;

- Decision to carry out complete dewatering of the mine;
- Development of an underground drainage network and start-up of the mine drainage program.

Figure 4.4 illustrates the dewatering of the water table in relation to time. The total volume of water removed is illustrated in figure 4.5.

Two extremely interesting points arise from:

1) The rock mass under production was depressurized (total stress) and dewatered.
2) The water table was lowered so that the overburden was dewatered to close to 20% saturation.

The mine dewatering resulted in an increase of the effective stress in the rock mass, thus improving the overall stability of the mine.

4.4.2 Soils overlying the mineralized zone

The nature of the soil overlying the orebody and the topography of the bedrock were identified in the following four manners:

1. Diamond drill holes were used to determine the topography of the bedrock. The majority of these holes were drilled in the hanging wall (figure 4.1) since they were primarily designed to intersect the mineralized zone at right angles.

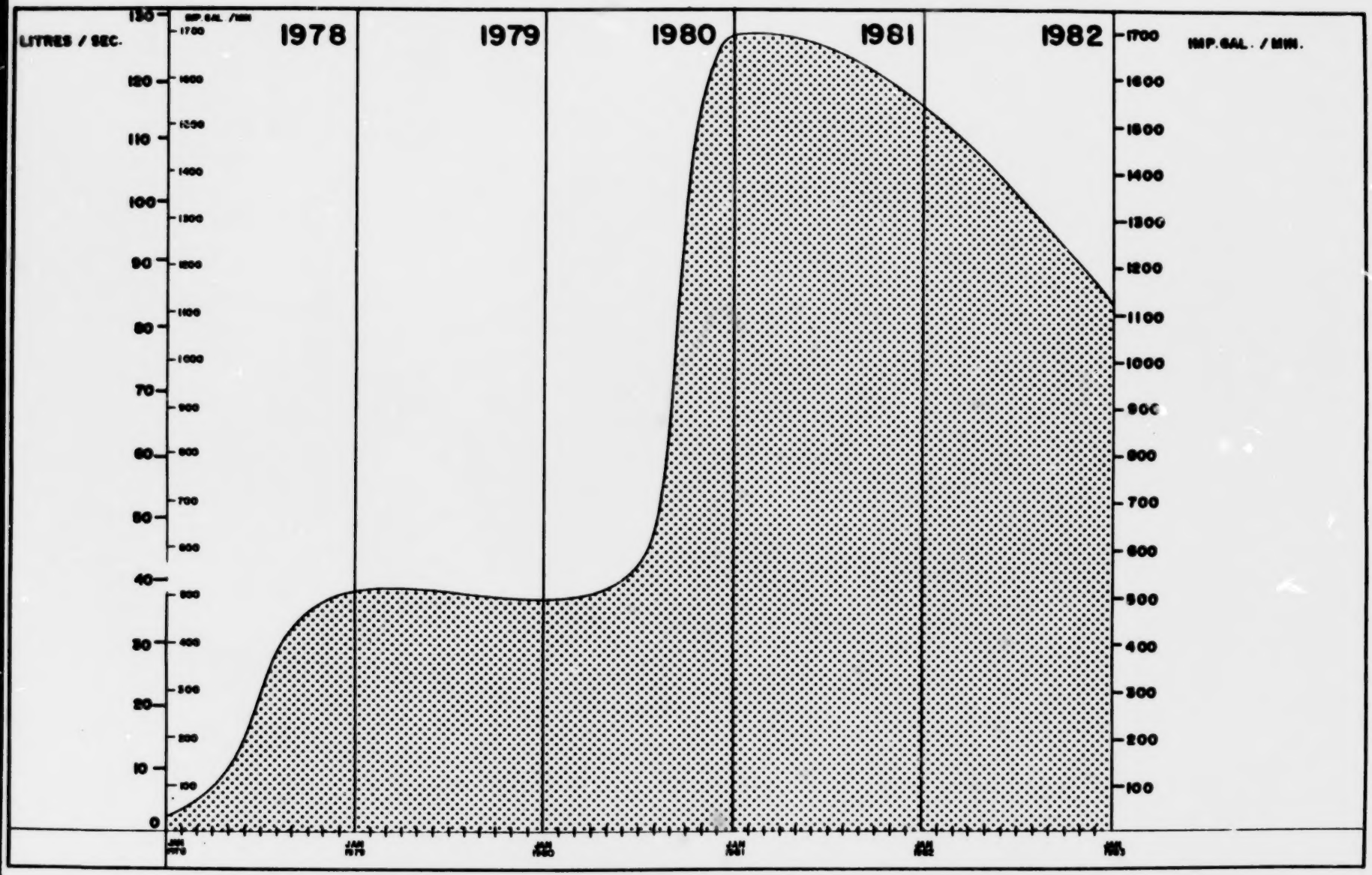

FIG. 4.5 VOLUME OF WATER REMOVED

2. During the realization of the hydrogeological studies, several holes were drilled to enable the evaluation of the layers of overburden and the determination of the location of the rock mass. Moreover, they were used to evaluate the behaviour of water in this zone.
3. Seismic surveys were conducted to locate the contact of overburden and rock mass.
4. Drilling was directed upwards from development drifts in the upper zone of the mine in order to better locate the point of contact while providing additional drainage of the overburden.

Information regarding the position of the contact of overburden and rock mass is continually gathered by the mine personnel, as illustrated in figure 4.6. The approach adopted is continuous since additional details regarding the geometry of the contact are constantly being provided by the results of exploration works. Data are gathered and compiled on all transverse geological sections at 10-metre intervals. This is a practical way in which to visualize the variations in elevation of the bedrock and to detect all significant depressions of the contact.

Laboratory tests were conducted in order to determine the mechanical properties of the principal layers of overburden, i.e. density, grain size, shear strength, cohesiveness, etc.

The management of the mine wished to examine the effects of drainage by directly studying the behaviour of free flowing overburden. They demonstrated that at the desaturation and depressurization level reached (σ total), the overburden can no longer flow because sufficient

water is not available to mobilize it. This test was conducted in a stope under development in which an opening was drilled between the roof of the stope and the contact of overburden 10 to 12 m higher.

Consequently, the surface pillar stability values should be adjusted as a function of the reduction in the vertical stresses which it must support.

4.4.3 MODEL OF THE MINE

A study on the stability of the openings of the entire mine was conducted in order to examine the influence of the following three parameters on the behaviour of the rock mass:

- variations in the competence of fill material
- variations in the thickness of the surface pillar
- variations in the equilibrium relationship as defined by Hoek and Brown (1980, Eq. 43).

The modelling technique consisted of an analysis program based on the finite element method. The objective of the program, in which boundary stress conditions were taken into consideration, was to determine the behaviour of the rock mass around the openings. The basic data gathered consisted primarily of the uniaxial compressive strength of the rock mass, calculated by applying the load point test to borehole cores, and the density of the material, Young's modulus and Poisson coefficient, calculated in a laboratory. The CSIR and NGI classifications were used to determine the two constants of the rock mass " m" and " s" as defined by Hoek and Brown in their equilibrium equations:

$$\sigma_1 = \sigma_3 + \sqrt{m\,\sigma_c\,\sigma_3 + S\,\sigma_c^2}$$ Eq. 43 (Hoek and Brown, 1980)

$$\sigma_t = 1/2\,\sigma_c\,(m - \sqrt{m^2 + 4S})$$ Eq. 45 (Hoek and Brown, 1980)

Where σ_1 = major principal stress at failure

σ_3 = minor principal stress applied to the sample

σ_c = uniaxial compressive strength of the intact rock material in the sample

σ_t = uniaxial tensile strength of the intact rock material in the sample

m and s = constants

These failure criteria were used in relation with the K_o factor defining the ratio of the horizontal stress to the vertical stress. The area is defined as isotropic, homogeneous and elastic in nature and bears uniform stress.

Within the framework of this study, it appears that K_o was estimated using values from 1 to 3. We do not know the value of σ_c used in this analysis. However, the average compressive strength of the intact rock of this mine is 80 MPa (11 600 psi); E = 5 800 MPa (7×10^5 psi) and V = 0.20 to 0.25.

This initial stability analysis provided the operators with indications on the probable behaviour of the rock mass and the location of potentially unstable zones.

4.4.4 Rock mechanics

This mine has assigned a special engineering team to implement a rock mechanics program. In order to obtain basic information to use in the previously described models, rock mass structures, including major discontinuities and rock types, were mapped.

The uniaxial compressive strength was calculated using the load point test conducted at various drilling sections in the footwall, hanging wall and mineralized zone. It is interesting to note that the value of σ_c varies considerably, ranging from 30 MPa (4 350 psi) to 160 MPa (23 200 psi).

The management plans to calculate the natural existing stresses in situ. An extensive area was mapped in great detail in order to enable an evaluation of the location of discontinuities within the rock mass and of the influence of these discontinuities on the stability of underground workings. At present, however, this information is unavailable.

4.4.5 Support

The support of the surface pillar is ensured by the overall stability of the mine, its own strength and various artificial support techniques. The desired minimum surface pillar thickness has not yet been reached and consequently, there is little to say regarding the support method employed. Nevertheless, the operators are already familiar with the properties of this sector and are conducting a series of tests in order to evaluate the type of support which will eventually be required. For

example, swellex bolts were tested in a highly oxidized, heavily fractured zone and grouted cables are used to support the wales.

The same method is employed as in the case of rib pillars left between stopes. Although the current standard specifies that these pillars must be 22 metres in width or length and 20 metres in height, the behaviour of the stopes must be taken into consideration. It is anticipated that the width of the pillars will be decreased by 50% in order to reduce the span of the roof to 11 metres under the surface pillar.

In these cases, a step-by-step approach is adopted in which the results of model simulations are either supported by the actual behaviour of the rock mass or adjusted according to a better characterization of the inherent properties of this rock mass.

4.4.6 Instrumentation and control

Due to the form of the orebody (see figure 4.1) and to the presence of potential zones of unstability, identified by means of mathematical modelling, the management of the mine decided to equip certain sections of the future surface pillar with instruments. Two multiple-point rod extensometers were installed from the surface, the first in the upper part of the hanging wall and the second in the surface pillar. These instruments enable the determination of displacement which may occur during the course of mining operations. They will be relocated along the longitudinal axis of the orebody, following the progression of operations.

In addition to these instruments, the stopes are also inspected regularly.

4.4.7 Geometry of surface pillars

As previously mentioned, the anticipated surface pillar has not yet been formed. At present, it is 35 metres in thickness and varies from approximately 13 to 45 metres in width. It is a slab of almost 500 metres in length resting on 22-metre wide stopes evenly spaced at intervals of 22 metres. This slab can be considered as a series of small surface pillars overlying the upper stopes. The underlying stopes will be completely filled. The dimensions of several stopes and overlying surface pillars are provided in table 4.1.

At this stage, we are unable to indicate the final shape of the surface pillar. However, figure 4.3 illustrates a section of the general aspect of the surface pillar.

4.4.8 Rock mass evaluation

Two geomechanics rock mass classifications are used by this company, that of the South African Council for Scientific and Industrial Research (CSIR) and that of the Norwegian Geotechnical Institute (NGI). These two techniques, which were applied throughout the mine, have their respective advantages and disadvantages and were used to identify and describe rock types in various transverse sections spaced at 90-metre intervals. Although these sections are judged to be useful by the operators, they are not sufficient for the detailed design of a specific stope. In order to obtain further details regarding the rock mass, the engineering team is considering increasing

TABLE 4.1 : DIMENSIONS OF SURFACE PILLARS AND UNDERLYING PILLARS

SURFACE PILLARS	EXAMPLES			
	No. 1	No. 2	No. 3	No. 4
OVERBURDEN:				
- Thickness (m)	45	45	45	45
SURFACE PILLARS:				
- Thickness (m)	35	35	35	35
STOPES:				
- Length (m)	30	48	22	22
- Width (m)	22	22	15	15
- Height (m)	20	20	20	20

Note: These pillars will be partially mined in order to leave no more than approximately 10 metres in thickness.

the number of this type of section.

Until now, these classifications have been primarily used for modelling of the mine, as mentioned in section 4.3.

4.4.9 GENERAL

We have already discussed the various studies conducted at this mine. The management of the mine has told us that they plan to simulate the mining sequence selected in order to verify its effect on the overall stability of the mine in general and surface pillar in particular.

Various options have been studied for the optimal production of the surface pillar to a level of -55 metres, leaving a crown pillar of approximately 10 metres thick (figure 4.6). The general layout considered by the management is as follows:

a. primary stope of 11 m
b. secondary stope of 11 m
c. pillar of 5.5 m
d. primary stope of 11 m
e. pillar of 5.5 m

4.5 ASSESSMENT OF THE VISIT

With the exception of matters on which the management has not yet made a final decision, we were able to gather a fair amount of information regarding the works realized at this mine.

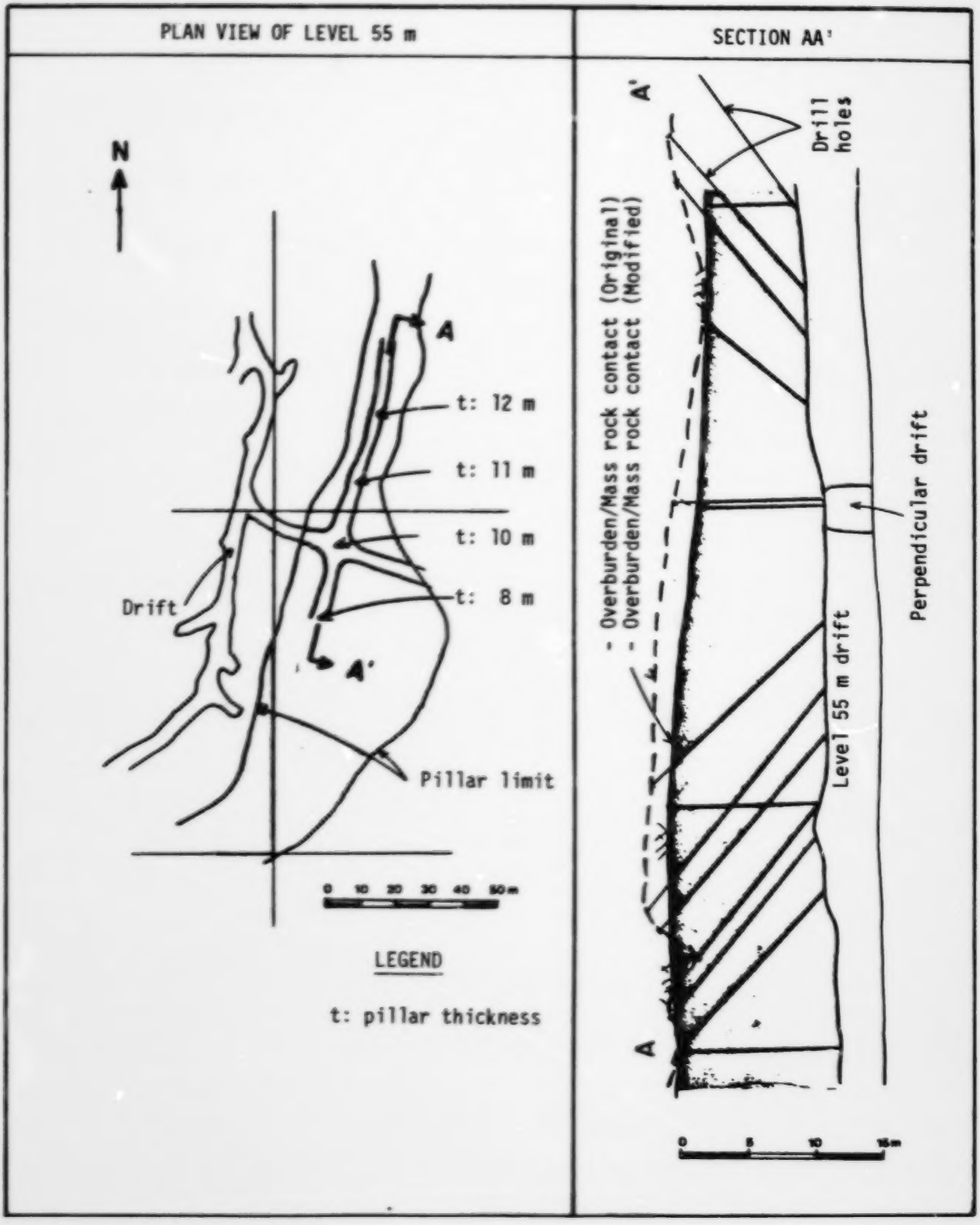

FIG. 4.6 LOCATION OF THE OVERBURDEN/ROCK MASS CONTACT

From our visit, it was apparent that the management has adopted a progressive and safe approach to the design of the surface pillar. Several factors are examined in order to obtain a set of guidelines, each factor contributing to the definition of the properties and behaviour of the rock mass.

The approach adopted by the management is also dynamic. The initial information gathered is supplemented by the experience acquired by the mine personnel during mining operations. The natural properties of the rock mass are modified by the drainage of the mined zone and must consequently be re-evaluated. Potentially critical zones are detected by mathematical model simulation and sections of the future surface pillar are equipped with instruments in order to enable the operators to determine whether or not a modification of the mining or support method is necessary.

Quantitative data of the geomechanic properties of the rock mass also supplement the qualitative information gathered at the mine site.

MINE 5

GOLD MINE

OPEN SHRINKAGE STOPING

VERTICAL RETREAT STOPING OR VERTICAL CRATER RETREAT (VCR)

AND CUT-AND-FILL MINING

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5.0 GOLD MINE OPEN SHRINKAGE STOPING, VERTICAL STOPING OR VERTICAL RETREAT STOPING OR VERTICAL CRATER RETREAT (VCR) AND CUT-AND-FILL MINING

5.1 BRIEF HISTORY OF THE MINE

This company has been mining low grade gold deposits since 1936. The area, which is approximately 3.2 kilometres in length, extends along the contact of two rock masses of differing geological ages, and follows the east/west longitudinal axis of an inverted syncline. Mineralization has occurred in volcanics, conglomerates and graywackes. The dip of the sedimentary strata varies from 70° to 80° N (figure 5.1).

More than 54% of the ore mined comes from the conglomerate rock of 12 to 18 metres in width. The graywacke formation located to the south produces 28% of the ore and the volcanic rock located further north provides 18% of the ore mined to date. Six to 18 metres of overburden composed of sand and gravel overlie the mineral deposit.

The ore is found in quartz veins and strockwork zones of variable thickness, generally positioned at N20°/45°E. They are located very close together and dip from 10° to 70° to the south. A major fault system plays a predominant role in the structure of the deposit. Hundreds of faults cut the deposit, producing horizontal and vertical displacements ranging from several centimetres to tens of metres.

The levels of mining are generally spaced at a distance of 61 metres and the principal shaft presently reaches a level of 945 metres.

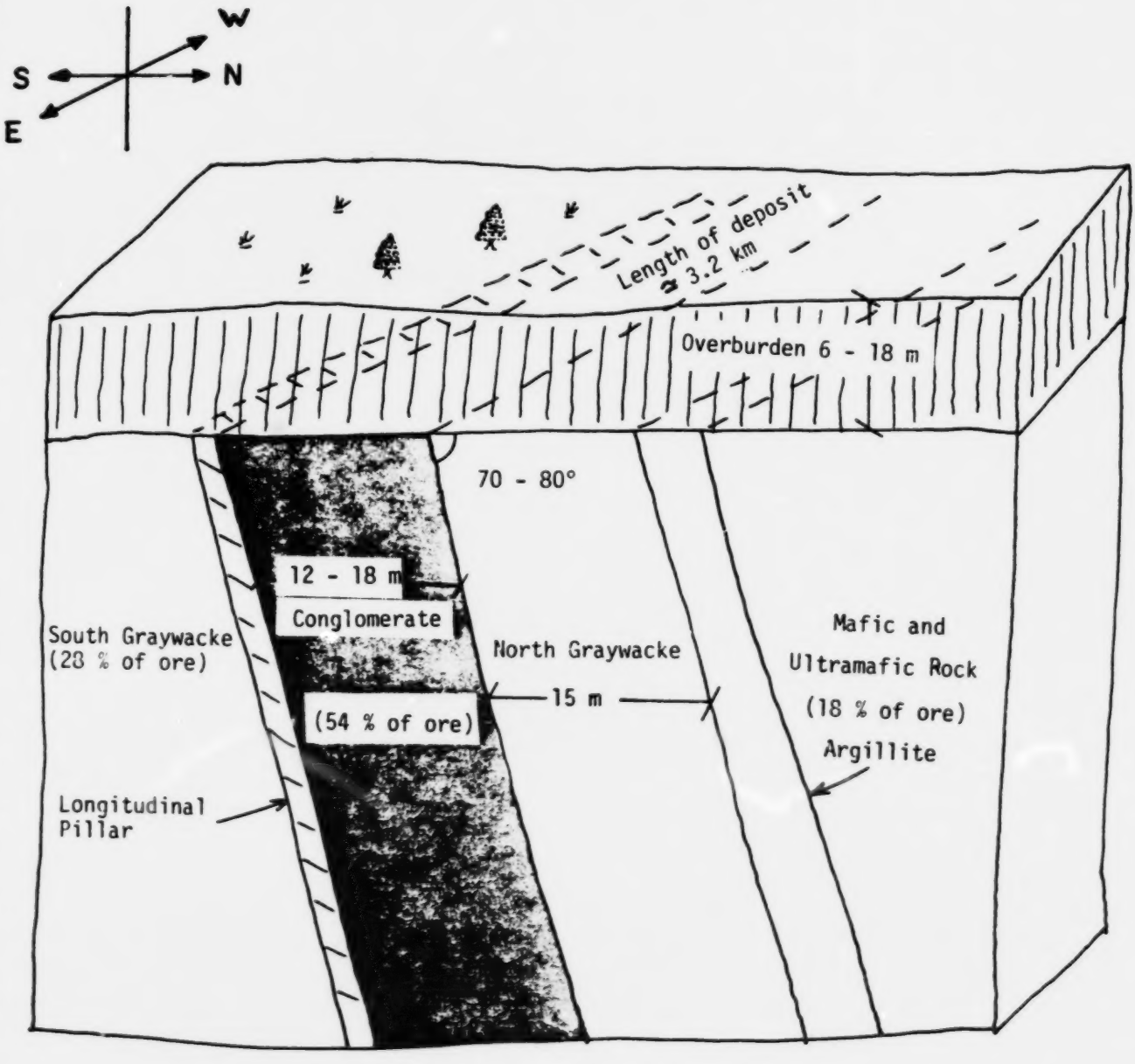

FIG. 5.1 CROSS SECTION OF THE MINERAL DEPOSIT

5.2 MINING METHOD

Shrinkage stoping was the initial method used to mine the ore from the upper stopes of the mineral deposit. Horizontal holes were drilled in the direction of the mineralized stratigraphic layer so that the ore could be mined in sucessive slices, starting from the bottom and advancing upwards. In some cases, near vertical blastholes were used instead. Part of the blasted ore was left in the stope where it served as a working platform for mining the ore above.

Upon completion of the mining of the ore body, the stope was abandoned without being filled. The stopes were separated by rib pillars in which manways were drilled. The dimensions of the rib pillars were as follows:

- Length : 15 to 24 m (depending on the width of the deposit)
- Width : 10 m
- Height : 53 m
- Distance between pillars : 58 m
- Pillars superposed from one level to the next

A longitudinal section of the upper stopes mined by open shrinkage stoping is illustrated in figure 5.2.

At present, the ore is being mined using both the vertical crater retreat (figure 5.3) and the cut-and-fill methods. In vertical retreat stoping, a horizontal slice of ore is excavated from the entire width and length (overcut) of the upper section of the stope, while draw-points are developed in the lower section.

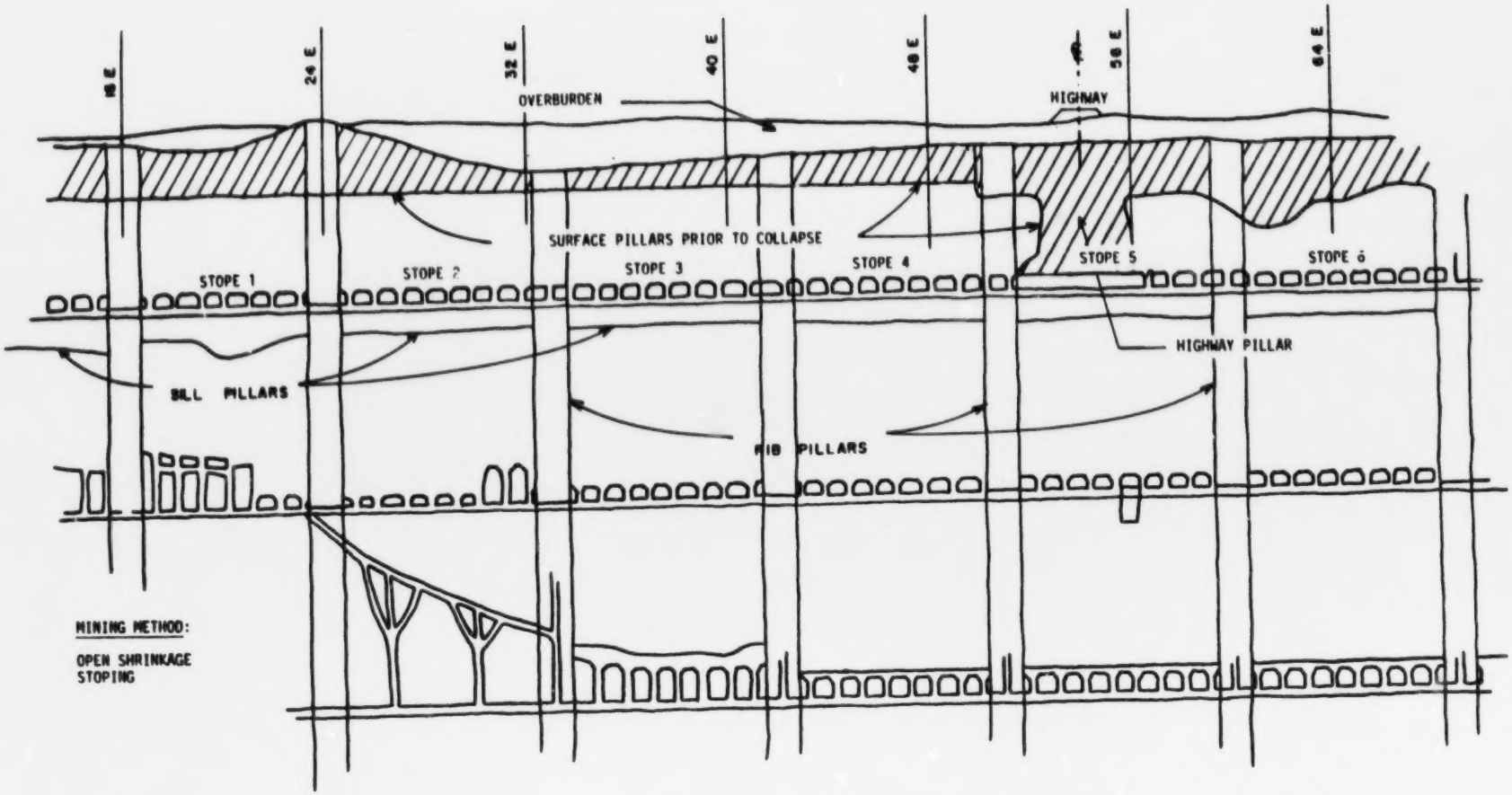

FIG. 5.2 SURFACE PILLARS AND STOPES: LONGITUDINAL SECTION

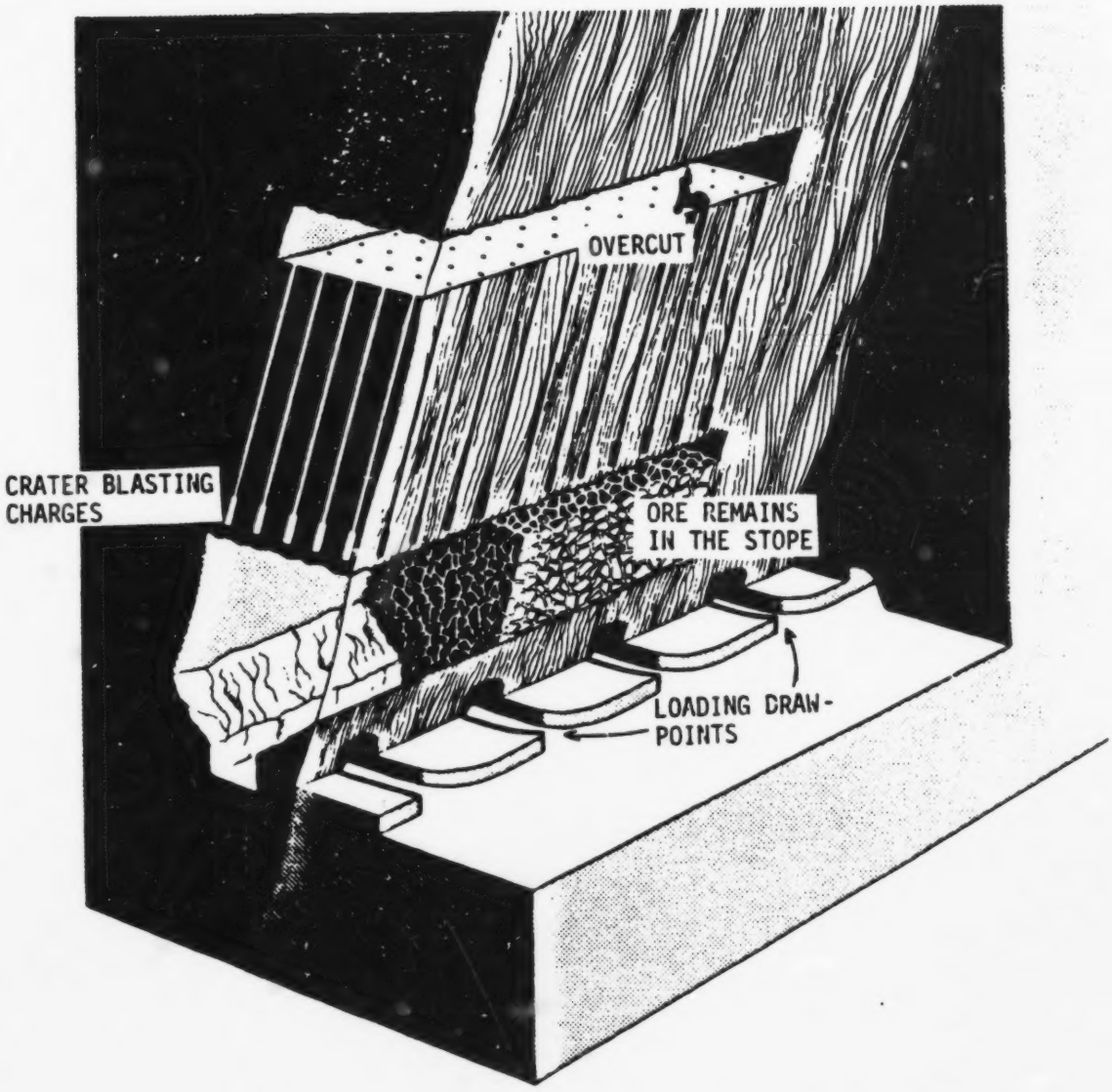

FIG. 5.3 MINING METHOD: VERTICAL CRATER RETREAT (VCR)
(Taken from Guide to Underground Mining, Atlas Copco, 1980)

Mining of the ore then proceeds as follows:

. 165 mm Ø holes are drilled starting at the top of the stope to the first slice of ore to be mined, that is, the slice situated immediately above the drawpoints.

. Subsequent slices are mined, starting from below and advancing upwards.

The dimension of the stopes mined using the VCR method are as follows:

. Length : 60 m
. Width : 15 to 25 m (depending on the width of the mineral deposit)
. Height : 53 m

As with shrinkage stoping, rib pillars delimit the stopes developed with the VCR method. These pillars have more or less the same dimensions as those previously mentioned and are superposed from one level to the next. At times, if the hanging wall is of poor quality, it may be essential to leave a pillar in the centre of the stope. Upon completion of mining, the stopes have not to date been filled with water-saturated sand or coarse-grained mill tailings (cemented or undemented) which have been passed through a cyclone. It must be noted that some of the centre and rib pillars may be recovered in the future if a good fill program can be incorporated with the mining. The cut-and-fill method, which has been described in previous sections, is also used for mining deeper stopes. The types of backfill material are the same as those to be used to fill stopes mined by means of the VCR method.

5.3 TERMINOLOGY USED

The management of the mine uses many terms interchangeably to designate

the same type of pillar. Rib pillars are sometimes referred to as end pillars, transverse pillars, and vertical pillars across the ore. Pillars that separate the upper stopes from the lower stopes are called sill pillars. They are horizontal and permanent and can also be referred to as crown pillars.

The term longitudinal pillar is used to designate a permanent pillar left between the conglomerate and the south graywacke (figure 5.1). This pillar follows the direction and dip of the contact of conglomerate rock and south graywacke. Pillars left in the center of some wide stopes are callec center pillars. Pillars located between the overburden and upper stopes are designated by the term surface pillar.

5.4 SURFACE PILLAR DESIGN CRITERIA

5.4.1 Study of surface and ground water

The management of the mine has indicated that no problems related to water which could hinder mining operations exist and that no hydrologic study has been conducted.

5.4.2 Soils overlying the mineral deposit

The surface pillars studied at this mine have all caved in and are now found in the stopes of the first level. Previously, more than 90% of the pillars were covered by sand and gravel deposits.

The thickness of these deposits varied from 0 to 18 metres, averaging approximately 11 m. No water was detected in the overburden. Following the collapse of the surface pillars several years ago, no

water entered the mine when the overburden gave way into the upper stopes. Further details of the collapse are provided in section 5.4.9.

5.4.3 Model of the mine

No computer model has been used in the study of the stability of the surface pillars.

5.4.4 Rock mechanics

A rock mechanics program has recently been set up by the management. Due to certain structural problems, it was decided to test various types of rock bolting. In order to achieve this, management has underlined the importance of detailed structural mapping. In addition to a description of drill cores (chich includes mineralization, host rock, the type of alteration, rock quality designation (R.Q.D.) and a study of jointing), structural characteristics such as shear zones or major and minor faults (including direction and dip) have been recorded, and the filling material of the discontinuities has been analyzed. This data will eventually be illustrated in plans and sections which will serve as guide to engineers responsible for underground works. Figure 5.4 illustrates a plan view of the first level in which haulage drifts and discontinuous contours (faults) mapped in these drifts can be distinguished.

General rock samples taken from the conglomerate rock have uniaxial compressive strengths varying between 105 and 207 MPa (15 200 and 30 000 psi).

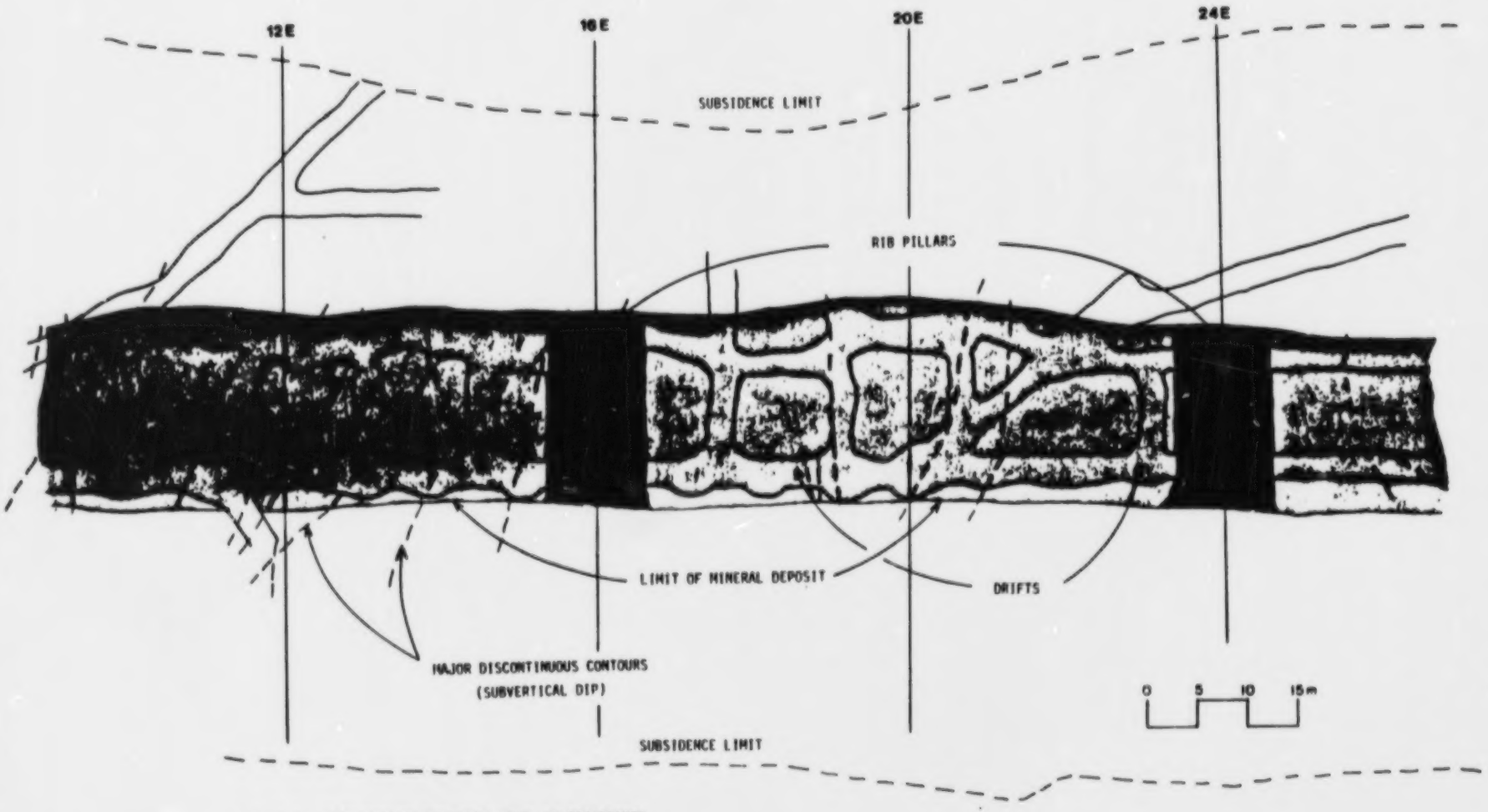

FIG. 5.4 PLAN VIEW OF THE FIRST LEVEL OF DRAWPOINTS

5.4.5 Support

In the past, the most commonly used methods of support included bolting, grouting (to a small extent in the stations), planking, meshing and steel arches. Today, support methods regularly used include bolts, resin-anchored or cemented reinforced steel bars, cables, Swellex friction bolts and split-set friction bolts.

As mentioned in section 5.2, some of the deep stopes are filled upon completion of ore excavation. This ensures greater stability of the stopes and prevents scaling of the walls. Furthermore, filling enables the recovery of rib pillars containing ore. As a protection measure, a large longitudinal pillar following the contact of conglomerate rock and south-end graywacke was left as waste in order to reinforce the foot wall.

5.4.6 Instrumentation and controls

All stopes are inspected visually in order to detect any irregularity representing possible instability or potential danger. Indicators are used to detect possible displacement of the rock mass; lines may be painted on both sides of a fracture in the rock or sheets of glass may be adhered to rock surfaces in order to show movement along a discontinuity. The distance between indicators located on either side of discontinuous contours is measured by means of a turnbuckle micrometer. By measuring the depth of drill holes, possible rock displacement can also be detected.

All pillars are periodically inspected, with special attention being given to those situated in weak zones.

5.4.7 Geometry of surface pillars

The dimensions of the six surface pillars and those of the underlying stopes are shown in table 5.1. The pillars themselves are illustrated in figure 5.2.

The surface pillars varied from 14 to 38 metres in thickness and the overlying overburden varied in thickness form 0 to 28 metres. Prior to the collapse of these pillars, a highway ran over stope No. 5. The surface pillar for this stope measured 27 metres in length, 17 metres in width and 37 metres in height.

The ratio of the thickness of surface pillars to the width of underlying stope was very low, varying between 0.8 and 2.2.

5.4.8 Rock mass evaluation

The rock mass can be evaluated using structural data gathered by means of drill cores and cartographic data. No classification scheme, such as " NGI" , " CSIR" or " LAUBSCHER" is used by the management. The quality of the rock mass can be evaluated using criteria developed by mining engineers and geologists. These criteria are based on their experience and reactions of the rock mass since the commencement of mining operations.

TABLE 5.1: DIMENSIONS OF SURFACE PILLARS AND UNDERLYING STOPES PRIOR TO COLLAPSE

Surface pillars	No. 1	No. 2	No. 3	No. 4	No. 5	No. 6
Overburden:						
. Thickness (m)	8	0-15	14	8	3	8
Surface pillar:						
. Thickness (m)	14	14	14	18	38	14-36
Stopes (mined by open shrinkage stoping)						
. Length (m)	50	56	58	58	59	58
. Width (m)	17	18	15	18	17	21
. Heigth (m)	37	37	37	37	37	37

5.4.9 General

The surface pillars were stable for slightly over 12 years before collapsing approximately 27 years ago. A cross-section of the collapse of the surface pillar to the first level is illustrated in figure 5.5. The mining method used to excavate ore from the upper stopes (open shrinkage stoping) left a series of empty stopes covered by thin surface pillars. It should be noted that the management of the mine had foreseen this collapse and that no injuries were sustained. The primary cuases of the cave-in of the surface pillars are the following:

- Underlying stopes were over-mined: changes in stress caused by the extraction of ore resulted in the collapse of the surface pillars which had become too thin (table 5.1).

- Stopes were left unfilled.

- Discontinuous contours were present in the rock mass (figure 5.4): the rock mass was characterized by a number of minor faults which probably accelerated the collapse of the surface pillars.

5.5. ASSESSMENT OF THE VISIT

The design criteria for the surface pillars applied by the management of the mine were developed based on the practical experience of the company's mining engineers and on examples taken from mines having similar characteristics to the mine in question.

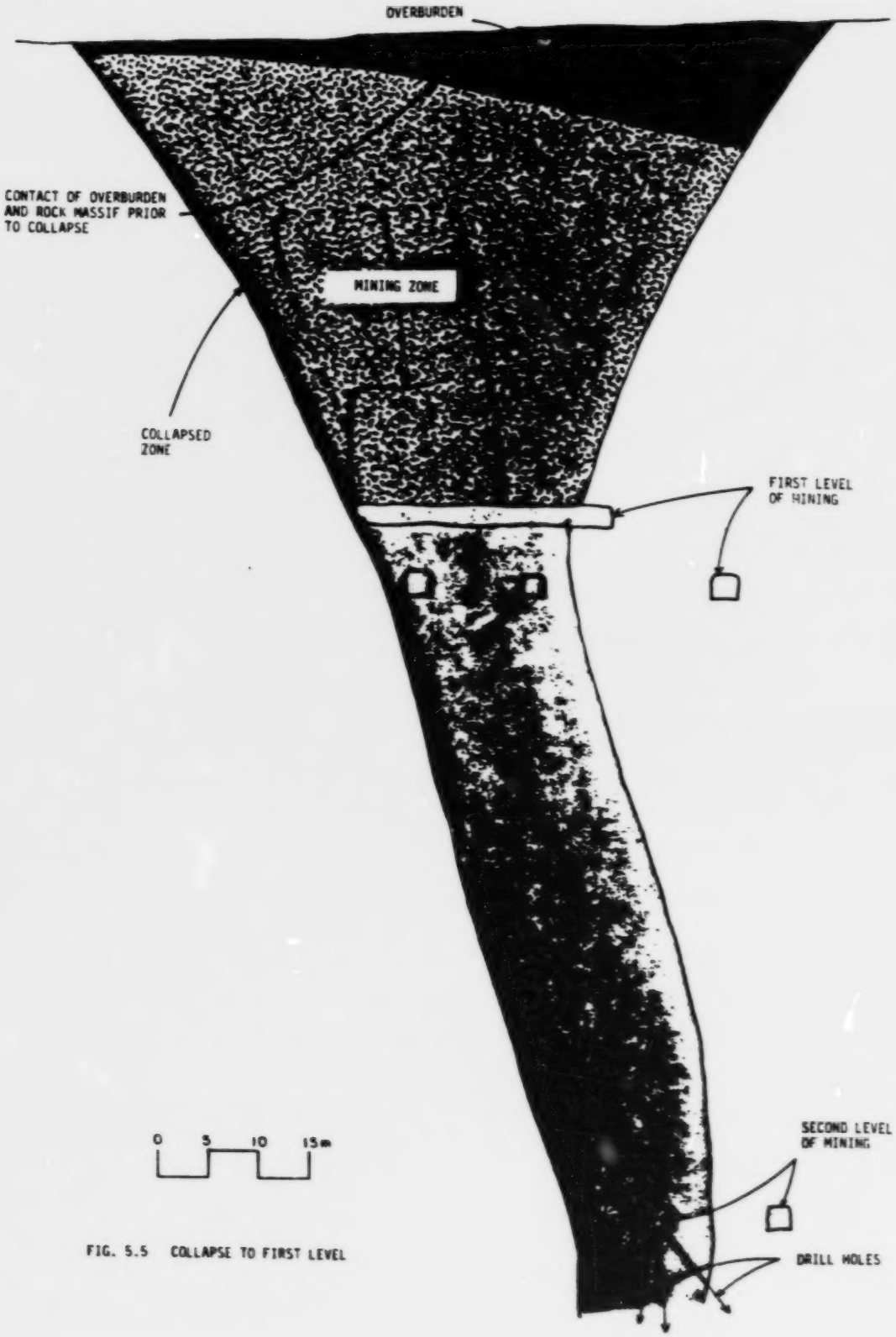

FIG. 5.5 COLLAPSE TO FIRST LEVEL

Management had quite accurately evaluated the dimensions of the upper stopes and pillars, for these remained stable for approximately 12 years after the closure of the stopes. They were aware of the nature of the overburden and of the impossibility that it could become saturated with water due to the lack of a substantial amount of water. In fact, it is this factor which led management to increase the level of mining of the upper stopes.

The management of the mine intends to use more and more field and laboratory data for the design of the various types of pillars. The mapping of discontinuities and the description of drill cores are systematically completed and studied by engineers responsible for underground works. The results of tests conducted on rock samples are integrated into analyses of the stability of the rock masses.

MINE 6

GOLD MINE: CUT-AND-FILL MINING, SHRINKAGE STOPING AND POST FILL METHOD

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6.0 GOLD MINE: CUT-AND-FILL MINING, SHRINKAGE STOPING AND POST FILL METHOD

6.1 BRIEF HISTORY OF THE MINE

Although the discovery of the mineral deposit was made in the early 1900s, it was not until 1964 and 1965 that intensive exploration works were carried out to determine the approximate dimensions of the orebody. At this time, the first exploration shaft was sunk. The property was recently re-evaluated and it was decided to begin development and production of the mine.

The mineralized zone under production is located in volcanic rock in which intrusions composed of peridotite, and to a lesser extent granodiorite, occur. The mineralized zone consists of breccia containing more than 5 pyrite, dipping between 20° and 50° west. The longitidinal axis of the orebody extends 250 metres with a north-south orientation; it is 500 metres in length (in the direction of its dip) and 20 metres in thickness. The breccia, which is very competent, cuts through the peridotites. In the contact area, the peridotites have undergone extensive alteration and have been transformed into talc and chlorite schists. The degree of alteration decreases the further one moves from the contact. The composition of the rock mass in which the orebody occurs, and the underground workings of the mine are illustrated in figures 6.1 and 6.2 respectively.

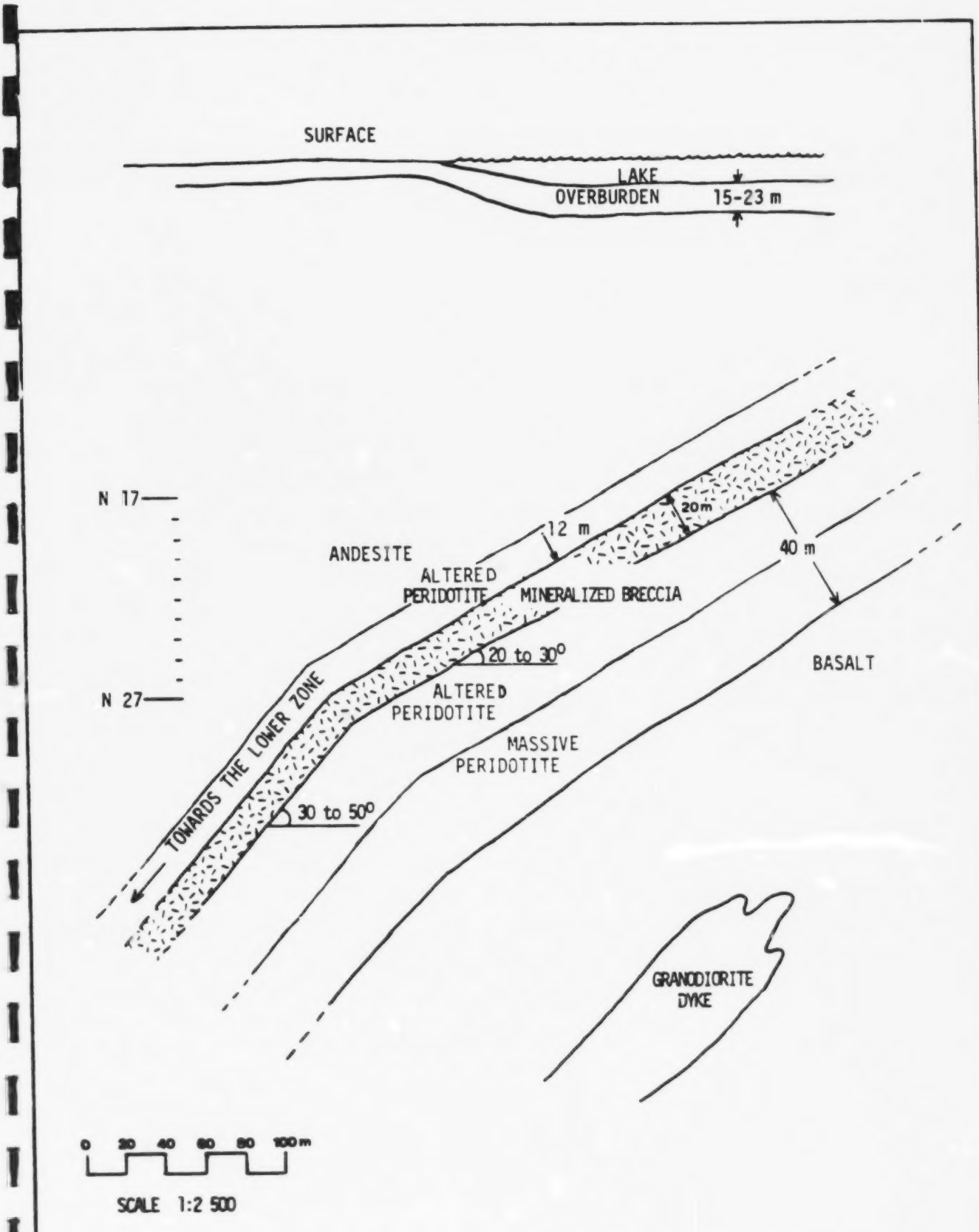

FIG. 6.1 ROCK MASS COMPOSITION (Simplified vertical section)

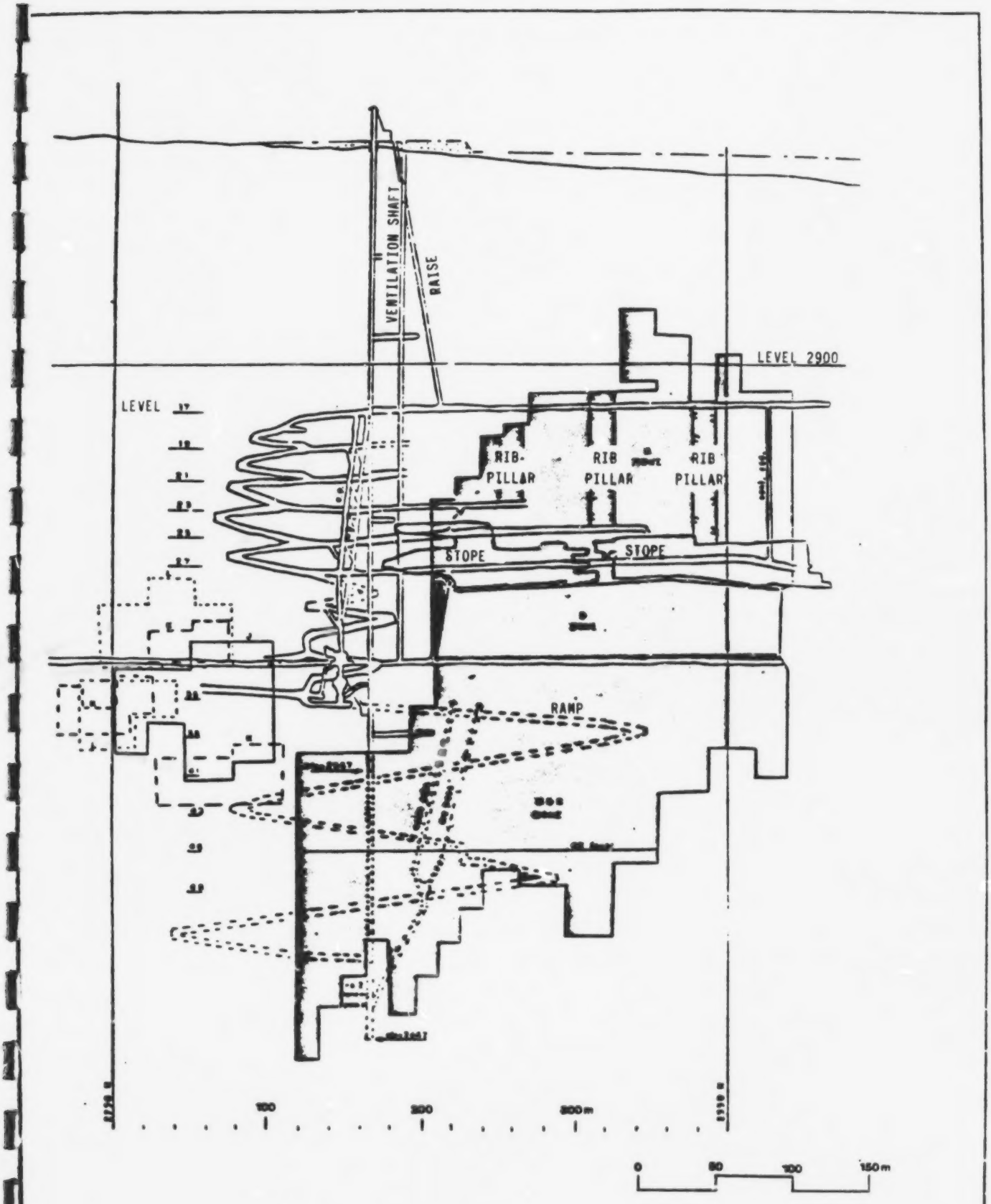

FIG. 6.2 LONGITUDINAL VIEW OF OREBODY AND STOPE AND LEVELS

6.2 MINING METHOD

The principal mining method employed is cut-and-fill stoping which accounts for 80% of total mine production. Open stope mining in which the stopes are subsequently filled, accounts for the remaining 20% of production. Access to the various sublevels, which are spaced at 20-metre intervals, is provided by a service ramp. At the hanging wall of each level, the drift branches into two or three drifts which provide access to the stopes. These drifts, which are 5 metres in width and 3.5 metres in height, intersect the orebody at right angles, thus enabling five slices, each 4 metres in height, to be excavated in each stope. After the ore is recovered from each slice, the stope is immediately filled. In certain zones, blasthole stoping is employed is order to excavate stopes measuring 15 m in width by 20 m in height. These stopes are also filled but only after all the ore has been drawn. The location of the drifts and stopes at one extraction level are illustrated in figure 6.3 and the mining method is outlined is figure 6.4.

The primary objective of the operators is to extract 100% of the mineral deposit. Primary mining operations account for the extraction of 60% of the estimated total volume. The filling of the stopes will enable the recovery of ore from the longitudinal pillars which represent 20% of the total volume. Then, depending on the behaviour of the rock mass, the last slice, which contains the remaining 20% of the ore, will be excavated from the three transverse pillars. This proposed sequence of mining operations may eventually be adjusted to account for stresses which are difficult to specify at this point.

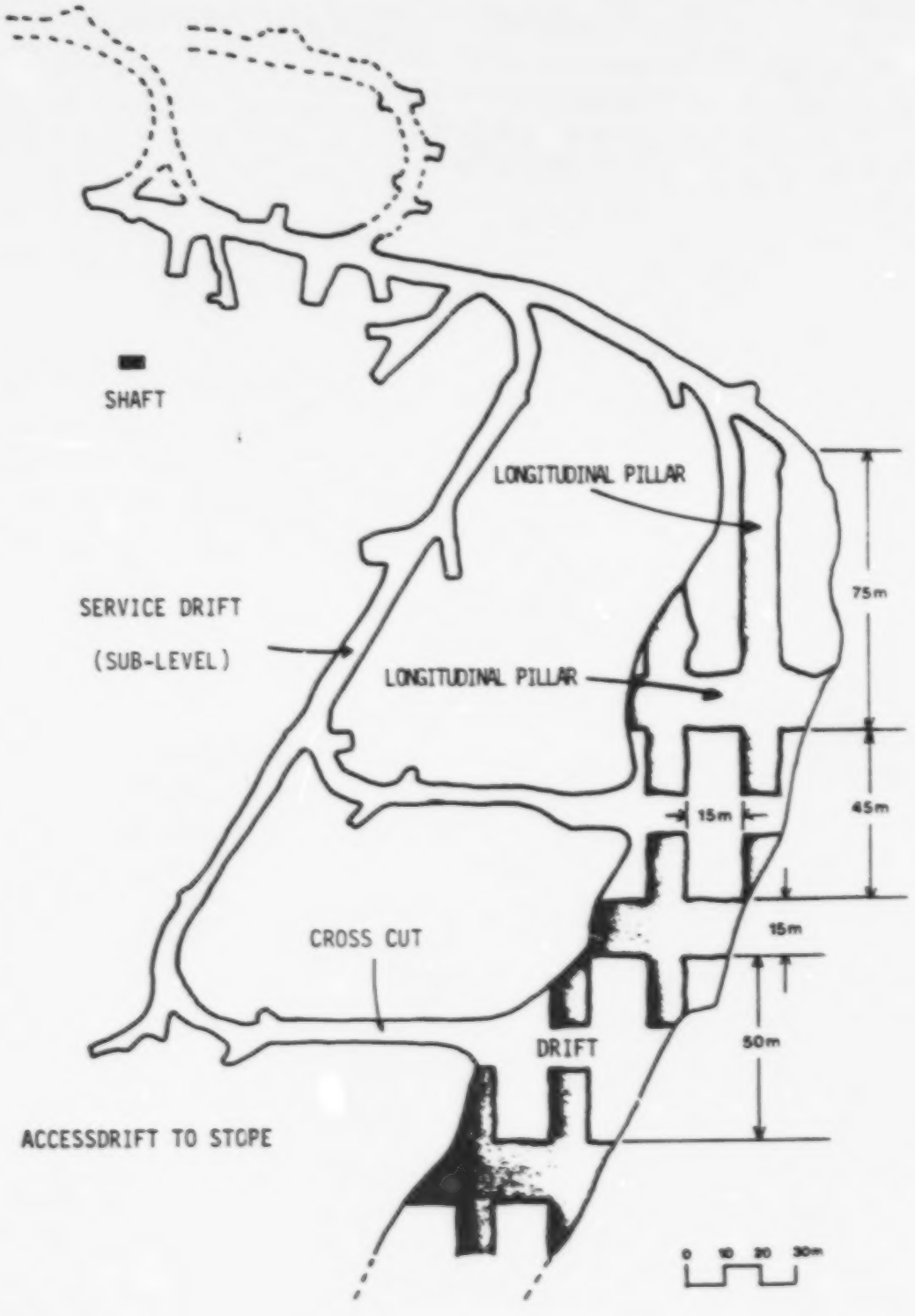

FIG. 6.3 PLAN VIEW OF A SUBLEVEL

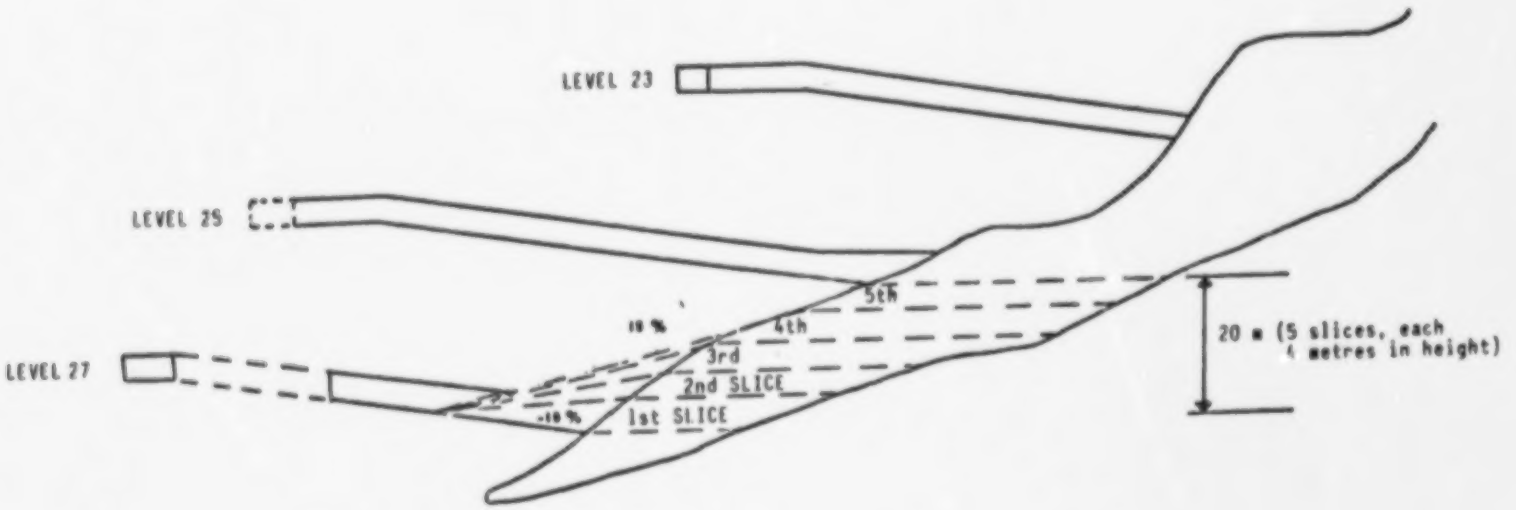

FIG. 6.4 DIAGRAM OF THE MINING METHOD

6.3 TERMINOLOGY USED

Due to the mining method employed, the management of the mine makes reference to only three types of pillars.

There are three transverse pillars, whose orientation corresponds to that of the dip. These pillars separate the footwall from the hanging wall. They are 15 m in width and their height corresponds to the thickness of the mineralized zone, or an average of 20 m.

The orientation of the longitudinal pillars follows the axis of the orebody. These pillars are left in a vertical position between the stopes and are not continuous. They measure on the average 10 m in width and 20 m in height. Backfill material will eventually be placed on three of their four sides. Their primary role is to limit the opening of extraction to widths allowing an appropriate support of ground.

The surface pillar is defined as the rock mass overlying the mineralized breccia. The definition provided by the management is similar to the one we proposed during the course of our study. It is difficult to determine the actual dimensions of the surface pillar in this mine (see figures 6.5 and 6.6) because the breccia is excavated to a level of approximately 110 m beneath the surface, it is relatively thin and measures almost 100 m in length along the horizontal plane, and its dip is variable.

6.4 SURFACE PILLAR DESIGN CRITERIA

6.4.1 Study of ground water and surface water conditions

The mineralized zone currently under production extends under a lake

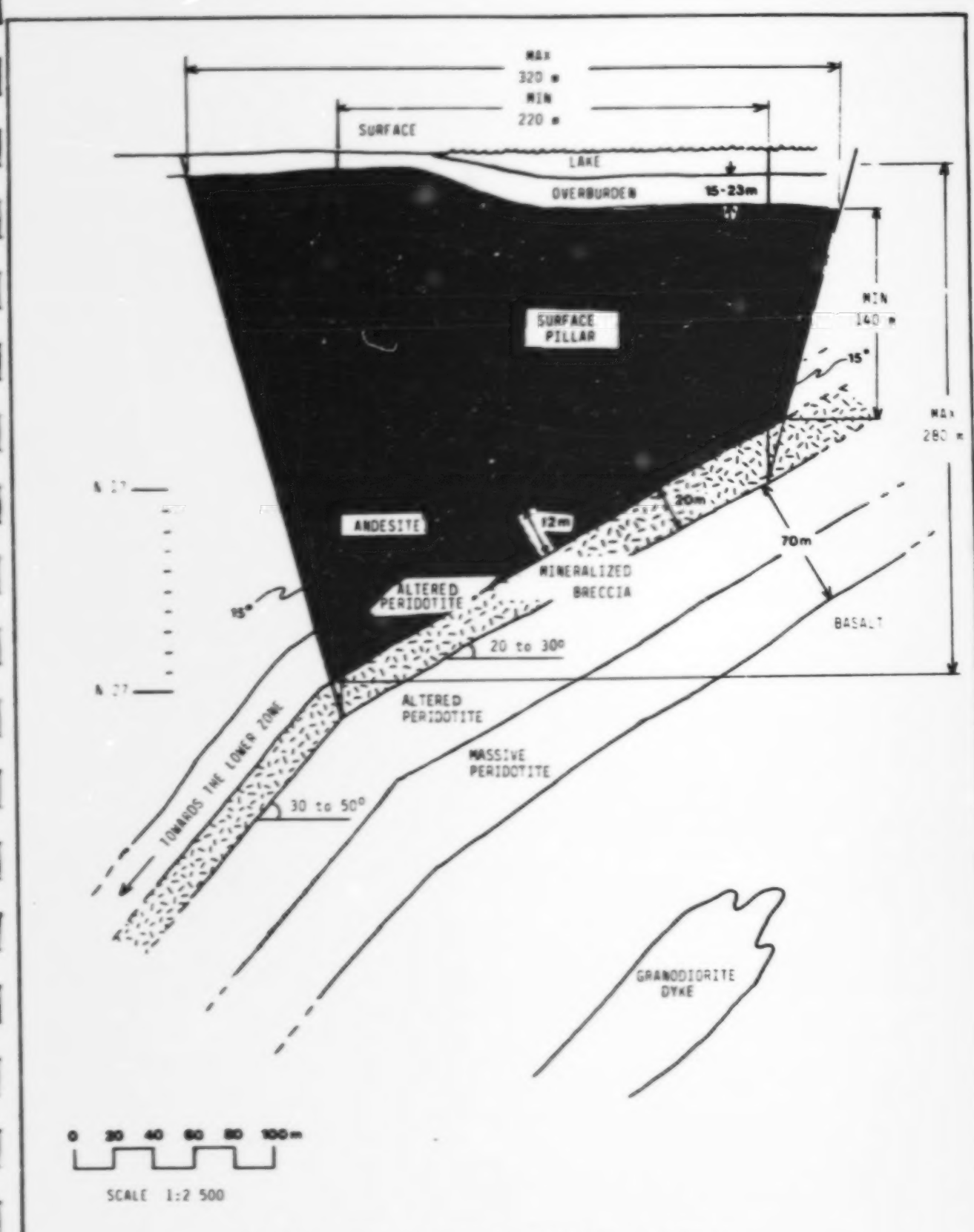

FIG. 6.5 VERTICAL SECTION OF THE SURFACE PILLAR

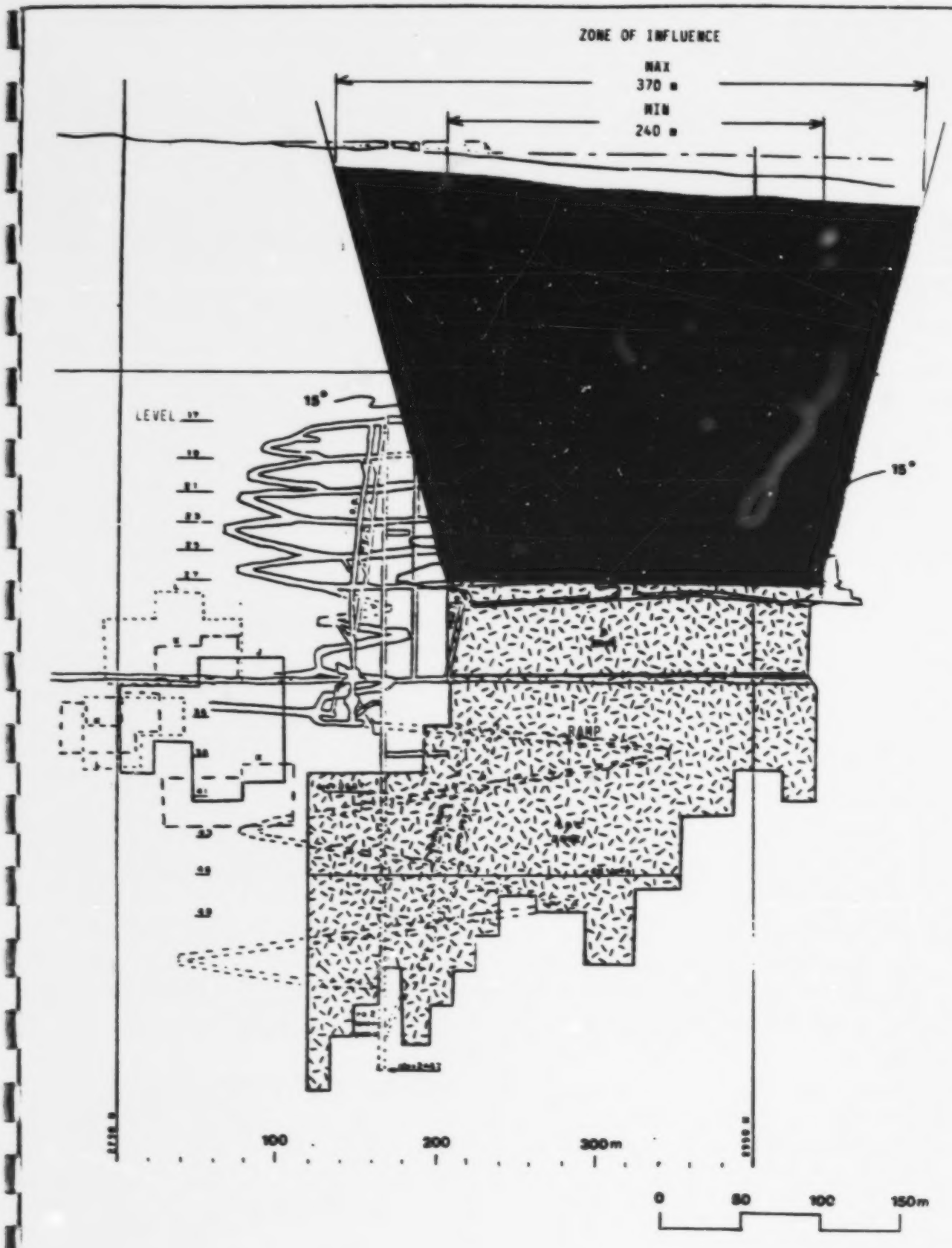

FIG. 6.6 LONGITUDINAL SECTION OF THE SURFACE PILLAR

which is 2 to 5 m deep. It should be noted that during the early stages of production, an exploration shaft situated at the west side of the mine was presumably filled with water. Very shortly after the dewatering of the present shaft, the old shaft dried up and has remained dry ever since. This suggests that current operations are sufficient to ensure drainage of the area and moreover, that there is no significant water supply within the mined zone. Since water is not a problem, the management of the mine has not conducted a hydrogeological study. It is believed that the clayey, silty nature of the layers of overburden at the botton of the lake prevent the vertical infiltration of water into the underlying rock mass. During our visit to the mine, we observed the sporadic permeation of small amounts of water in the contact zones of altered peridotite and the virtual absence of water in the mineralized breccia. Moreover, the hanging wall was completely dry. For these reasons, piezometers have not been installed.

6.4.2 Soils overlying the mineralized zone

The overburden varies from 15 to 23 metres in thickness and is composed of a bed of clay, of stratified silt and of sandy till. These beds are of variable thickness.

6.4.3 Modelling of the mine

The general stability of the mine was evaluated by means of computer model based on the finite-element method and applied to a typical transverse section of the mine. First, several vertical slices representative of the mined-out zone were selected. Because the model employed was a two-dimensional model, these slices were sufficient. Local geological data, obtained from holes drilled from

the surface during the initial exploration phases and from underground exploration and development drifts, were compiled. Borehole cores were re-evoluated and the rock quality designation was determined for the entire rock mass. Only major structures were included in the model, which generally corresponded to a change in rock type.

The five principal types of rocks found in the zone under production are listed in figure 6.1. They will be described in greater detail is the following section.

The two-dimensional model uses the following basic data:

- uniaxial compressive strength
- rock density
- Young's modulus
- Poisson's ratio
- natural vertical and horizontal stresses
- cohesiveness (c) and angle of internal friction(s)

In this case, the model was defined as elastic-linear and the failure criterion employed was that of Mohr/Coulomb. The average laboratory value of uniaxial compressive strength obtained for each type of rock was adjusted before it was used in the model. In order to take the size effect, shape effect and presence of discontinuities in the rock mass into consideration, factors which were predetermined by the user were employed. The theoretical uniaxial compressive strength value was reduced by factors ranging from 75% for the peridotite to 10% for the competent breccia. The compressive strength value used for the basalt and andesite was 70 to 75% of their laboratory values. The average uniaxial compressive strength of these rocks is listed in table 6.1.

TABLEAU 6.1: UNIAXIAL COMPRESSIVE ROCK STRENGTH

MATERIAL	SOURCE OF SAMPLE	UNIAXIAL COMPRESSIVE STRENGTH MPa
Andesite	Hanging wall	99
Altered peridotite	Contacts	43
Breccia	Orebody	71
Basalt	Footwall	185
Granodiorite dyke	Local	233

The sequence of mining operations was not simulated with this model.

The interesting aspect of this model is that it enables the management to concentrate on potentially weak or unstable zones by identifying areas in which it would be advantageous to use reinforcement systems and to install instruments for measuring displacement in the near future. The model actually serves as a guide for the planning of the mining sequence.

As previously mentioned, the model was applied to the mine as a whole. This approach was adopted a second time to study the upper zone of the mine, or the zone which has been defined as the surface pillar. The management feels that the results obtained in the second study are more or less the same as those obtained when the model was applied to the mine as a whole because the zone in question is of considerable thickness and because the same basic data were used.

6.4.4 Rock mechanics

The company has been interested in rock mechanics since the earliest stages of operation. In addition to the head engineer, a rock mechanics specialist has been working full time in this field. There appears to exist close collaboration between the rock mechanics specialists and the geology department of the mine. Rock mechanics data were obtained from borehole cores which indicate the presence of five specific zones of rock and from laboratory tests which were conducted in order to describe the properties of these rocks and to evaluate their behaviour. All borehole cores are evaluated for they give further information regarding these zones. The importance of the structural aspect

can thus be determined and the rock quality designation can be obtained in a systematic fashion. In situ measurements are then taken in order to determine the values of the natural horizontal and vertical stresses. With these values, the K_o ratio, defined as $\frac{\sigma h}{\sigma v}$, can be calculated.

According to the results obtained, K_o equals 3.0 in the direction NE-SW or along the longitudinal axis of the orebody while at right angles to the deposit, that is in a NW-SE direction, K_o varies between 1.5 and 2.5

This information is used by the department in the planning of mining operations and local support systems and in the evaluation of the general stability of the mine. Furthermore, various instruments are installed at the mining site to monitor local conditions.

6.4.5 Support

The principal method of controlling stope stability is through the use of either temporary or permanent support pillars. As previously mentioned, the orebody is intersected by three transverse pillars, each measuring 15 metres in width and the stopes are separated by longitudinal pillars measuring 10 metres in width. Moreover, the stability is ensured by the hydraulic filling of the stopes. The fill is mixed with cement to produce a harder surface; in regular cut and fill stopes, the ratio is 30:1 whereas in zones which have been mined by blasthole stoping or open stope mining, the ratio of fill to cement is 20:1. The fill material serves as a working platform for the mining of subsequent ore slices and prevents excessive or sudden displacement of the stope walls, hanging wall and footwall.

The support of the development drifts and permanent adits is ensured by the installation of resin-anchored bolts sometimes combined with 10 cm X 10 cm mesh. The bolting pattern generally employed is 1.2 X 1.2 metres. The length of the bolts varies, depending on the size of the openings and nature of the rock.

The most unstable or potentially unstable zone, identified by experience and modelling, is situated in the hanging wall, at the contact of the orebody. As illustrated in figure 6.7, it is composed of altered peridotite and is approximately 12 metres thick, on average. The support system consists of a series of grouted untensioned steel cables installed at right angles to the peridotite formation. These cables are installed from the first stopes mined, crossing the mineralization and penetrating at least 10 metres into the peridotite. This type of support system appears to be very effective. The operators are able to verify the installation of the cables during excavation of the ore slice in the stope located immediately above. The steel cables are 2 cm in diameter and vary from 12 to 21 metres in length. The installation pattern is 2.5 X 2.5 metres. Regular cement, type 10 or 30, is used.

Regular expansion shell bolts are used in the stopes to ensure the safety of the work teams. They are of temporary usefulness since the stopes are subsequently filled.

6.4.6 Instrumentation and control

As previously mentioned, the hanging wall is a probable zone of unstability and is systematically bolted by means of grouted steel cables.

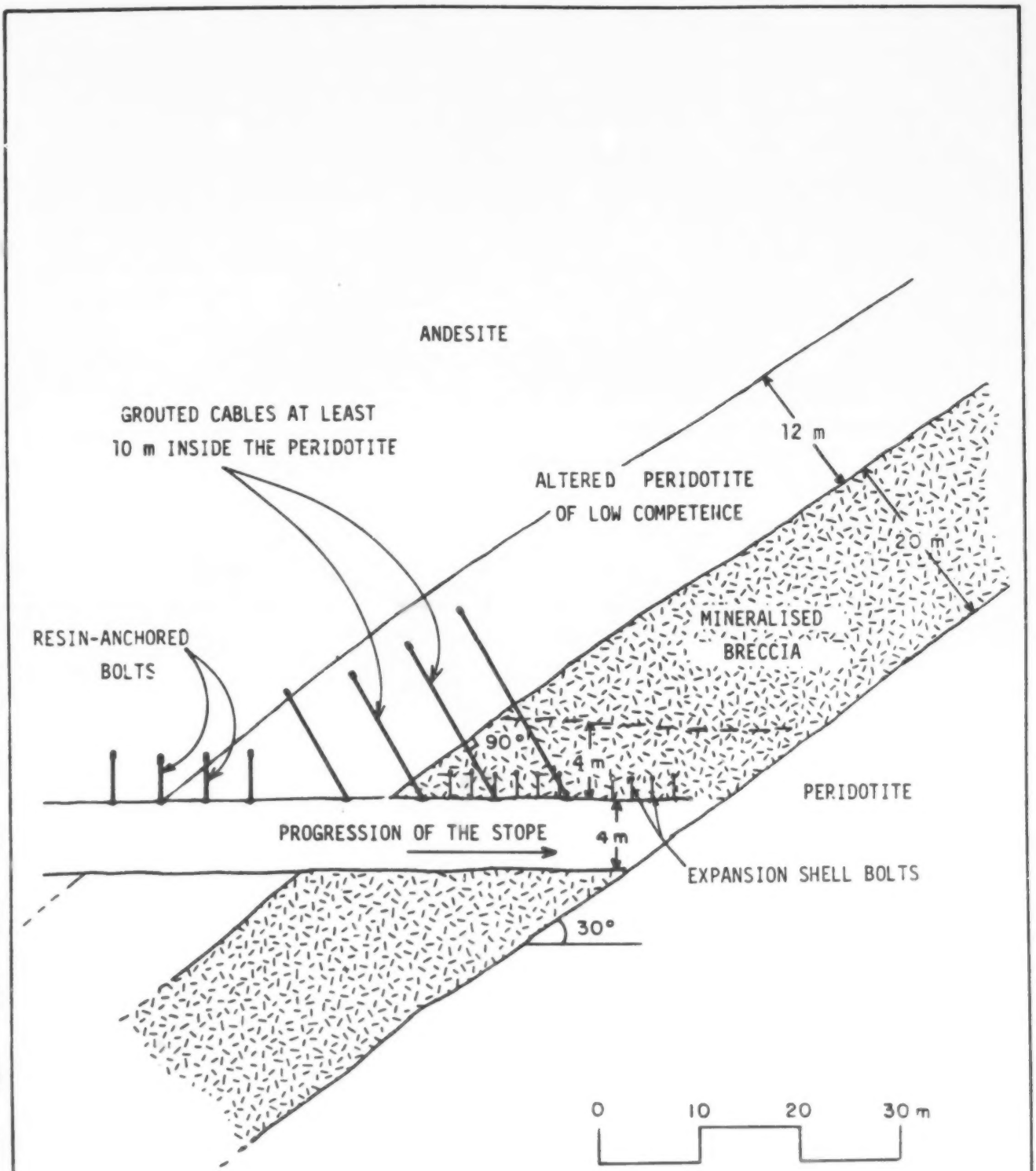

FIG. 6.7 EXAMPLE OF SUPPORT SYSTEMS

In order to obtain data regarding the behaviour of this zone, seven rod extensometers have been installed, several of which are located at level 17, inside the surface pillar. These instruments were anchored at 4 or 5 points. To date, no displacement of the hanging wall has been detected. These instruments constitute an important element of the company's rock mechanics program.

Stability is also evaluated locally by routine inspections of the stopes under production. The mine uses the heads of two extensometers as reference points for the measurement of the convergence of a particular drift in the upper level of the mine. One extensometer is installed in the floor of the drift and oriented towards the overbody; a second extensometer is installed in the roof and oriented upward. Both instruments are located in the same vertical plane. The two reading heads are used to determine whether or not convergence of this drift has occurred.

6.4.7 Geometry of the surface pillars

It is very difficult to determine the three principal dimensions of the surface pillar, as is seen by referring to figures 6.1, 6.5 and 6.6. Due to the presence of the three transverse pillars, we can consider there to be three surface pillars situated immediately above the upper stopes. Because of the weak dip of the upper zone of this orebody, we are inclined to include the entire hanging wall in our definition of surface pillar. Therefore, the pillar has the dimensions illustrated in figures 6.5 and 6.6 and listed in table 6.2.

TABLE 6.2 : DIMENSIONS OF THE SURFACE PILLARS AND UNDERLYING STOPES

SURFACE PILLARS	EXAMPLES			
	No. 1	NO. 2	No. 3	No. 4
OVERBURDEN:				
- Thickness (m)	19	19	19	19
SURFACE PILLARS:				
- Thickness (m)	179	179	179	179
STOPES:				
- Length (m)	250	250	250	250
- Width (m)	40	40	45	40
- Height (m)	24	24	24	24

NOTE All data listed represent average values.

6.4.8 Rock mass evaluation

As mentioned in section 6.4.3, the evaluation of the rock mass was based on exploration borehole cores drilled at the surface and from available underground drifts. In addition to the preparation of traditional plans and sections, considerable emphasis was placed on determining the rock quality designation (R.Q.D.) of the rock mass, since the re-opening of the mine.

The management was unable to obtain a clear image of the rock mass by analyzing the distribution of the R.Q.D. values of the five rock classes; due to the frequent change in classes, it was impossible to delineate continuous zones or link drill holes together. Therefore, they decided to limit the number of rock classes to three, to identify each of them with a different colour and to qualify each class.

The results of this simplification of the general method is as follows:

R.Q.D. RATING	COLOUR	ROCK DESCRIPTION
0 - 50	red	Very poor/poor
50 - 75	yellow	Medium/good
75 - 100	green	Very good/excellent

The following table represents Deere's original classification (1964):

R.Q.D. RATING	ROCK DESCRIPTION
<25	Very poor
25 - 50	Poor
50 - 75	Fair
75 - 90	Good
90 - 100	Very good

Although the results obtained at this mine cannot be compared with those obtained at other mines, the simplification of the classification technique enabled the mine personnel to visualize in vertical sections the relative position and class of the various rock zones. Moreover, based on this information combined with the mapping of developed exploration drifts, the management was able to determine the location of the drifts, adits and service installations.

The rock quality designations obtained using the simplified system are sufficient in that they adequately describe the rock mass and meet the needs of this particular operation.

The company now systematically conducts this test on all borehole cores drilled. No other rock mass evaluation technique is used on a routine method.

6.4.9 General

Due to the tabular form of the orebody and the slight dip of its upper section, the management of the mine has recently been considering evaluating the effect of the mining sequence on the overall stability of the mine.

6.5 ASSESSMENT OF THE VISIT

This property is uncommon in that we must consider the surface pillar as a large zone of rock mass overlying the upper stopes of the mine. The hanging wall is therefore an integral part of the surface pillar whose dimensions are considerable.

This company has adopted a dynamic approach which is constantly adopted to the progression of mining operations. The stability of the surface pillar is directly related to the overall stability of the mine. Particular zones of low competence are delineated by the application of models based on the finite-element method. The models correspond to reality to a greater extent with the constant addition of new field data.

POOR

QUALITY

ORIGINAL